ENGINEERING AND MINING JOURNAL

DECEMBER, 1941

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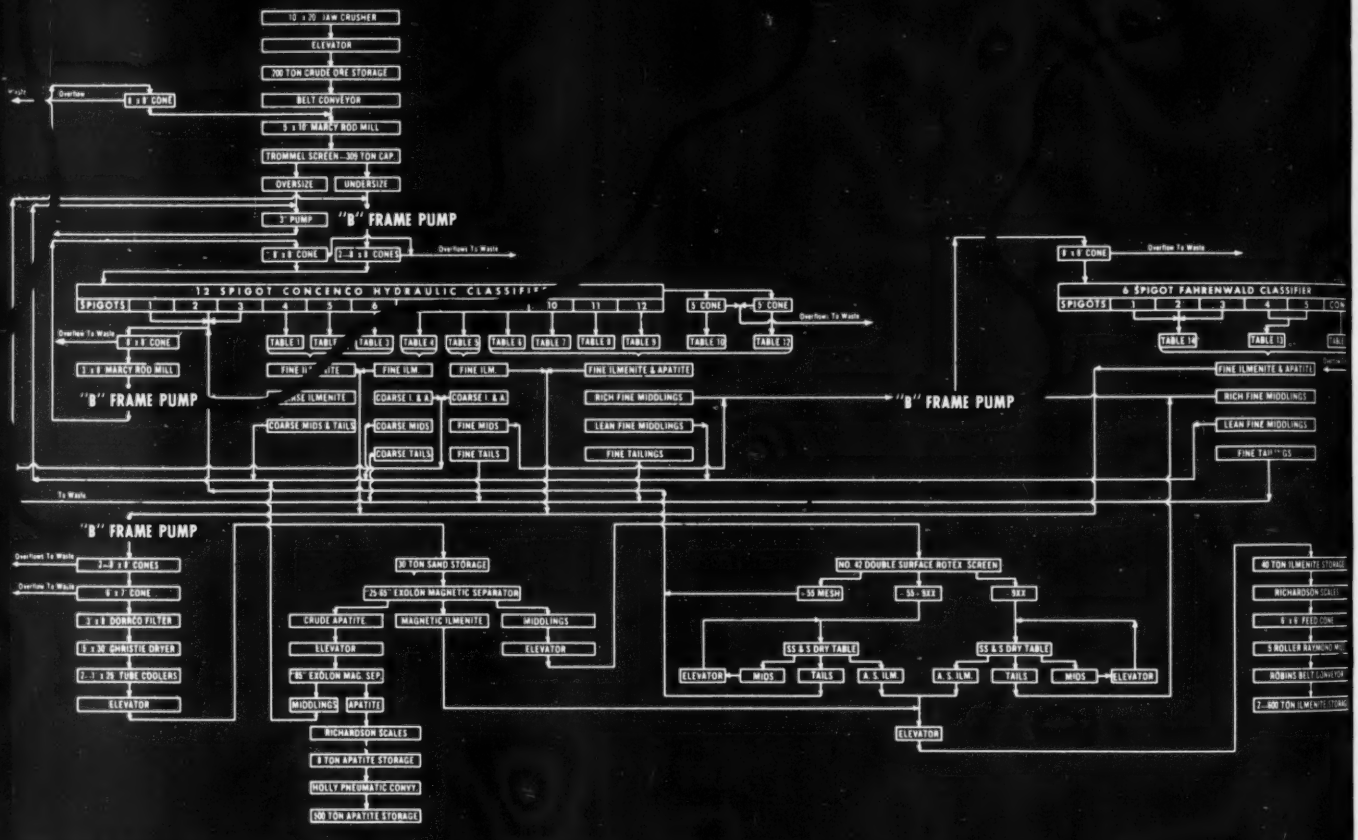
NEW INSTALLATION
Figures Not Available

.0011¢ PER TON
B-Frame Hydroseal Slurry Pump handling Concentrates, all minus 35 mesh. Pumps 73 tons of solids in 24 hours at 6.6% solids. During 16½ months this Hydroseal handled 34,821 tons at a repair parts cost per ton of .0011¢

.00345¢ PER TON
B-Frame Hydroseal Slurry Pump handling Rod Mill Discharge. Pumps 159 tons of solids in 24 hours at 21.6% solids. During 477 days this Hydroseal handled 75,843 tons at a repair parts cost per ton of .00345¢.

.0044¢ PER TON
B-Frame Hydroseal Slurry Pump handling Middlings, all minus 14 mesh. Pumps 35.5 to of solids in 24 hours at 5.9% solids. During 16½ months this Hydroseal handled 15,979 tons at a repair parts cost per ton of .0044¢.



THE repair parts cost per ton of these three Hydroseal Slurry Pumps is typical, for conditions such as those described. Altogether, during the 16½ months, 126,643 tons were handled at a total repair parts cost of $371.39 or an average cost per ton of .0029c. If you operate pumps under similar conditions, compare your repair parts cost with these. If your conditions are different, write us describing them. Our "Case History" File of Hydroseal Pump Installations undoubtedly lists pumping costs, under conditions like yours. We will gladly tell you, so that you can compare them with your costs. We can't always tell you the name of the installation, for some of our customers ask us not to do so . . . If you do not have the Hydroseal Pump Catalog No. 140 or the Packless Acid Pump Catalog No. 741, write the nearest office listed below.

PACKLESS ACID PUMPS AND

HYDROSEAL

***MAXIMIX* RUBBER PROTECTED — SAND AND SLURRY PUMPS**

Attention is called to the possibility of applying mineral-dressing processes to the reclamation of metals and alloys from industrial wastes. In the January issue W. E. Keck and F. A. Paciotti will cite research done on drosses, dusts and sludges in the laboratory of Michigan College of Mining and Technology, at Houghton. Particularly they will describe their investigation of a grinding sludge—a comprehensive study that has been carried through to satisfactory commercial operation.

▼

Iron mining is active on the Menominee range, in the Northern Peninsula of Michigan, with a dozen or more underground properties in operation. The ores of this part of the Lake Superior Iron Country are high in silica and phosphorus, a circumstance that ordinarily tends to limit their market. Current mining practice on the Range will be summarized in the January issue.

▼

Next month also the following: the diesel-electric plant at the new fluorspar-zinc-lead mine of the Mahoning company in southern Illinois, making current at 240 v., a.c., with a survey of the motors and drives in the mill—by J. H. Edwards; a shaft-sinking job in the Philippines—by J. C. Alexander; the Potash Company of America's treatment plant at Carlsbad; and more about placer valuation in Alaska.

▼

49th State. Determined miners with pistols in their belts reflecting the spirit of '49 were recently reported to be stopping motorists near the California-Oregon border to give out handbills advertising the region's mineral wealth and demanding better roads to permit development of these resources. To lend weight to this protest the residents of the four counties involved are said to be "seceding" each Thursday and calling themselves the 49th State of the Union, "The State of Jefferson."

H. C. PARMELEE
Editor

ENGINEERING AND MINING JOURNAL

H. W. CLARKE
Vice-President

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COVER ILLUSTRATION: Dumping molten matte into a converter at the Oroya, Peru, smelter of the Cerro de Pasco Copper Corporation.—Photo by Jules Bucher, Three Lions.

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From Bullets to Battleships

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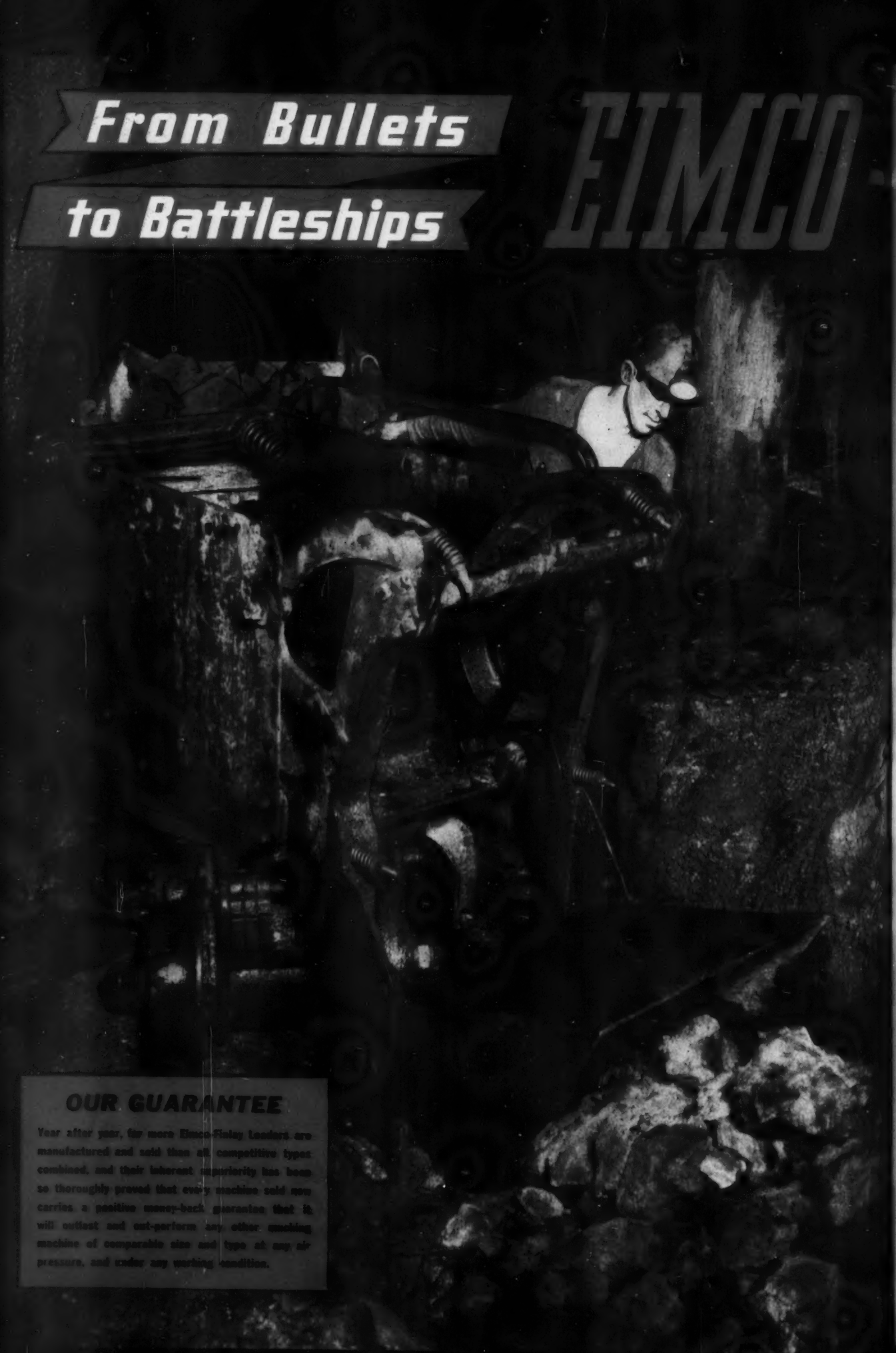

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E. & M. J. Goes to South America

WHEN the First Pan American Congress of Mining Engineering and Geology convenes in Santiago, Chile, on January 11, 1942, *Engineering and Mining Journal* will be represented in the person of John B. Huttl, assistant editor, whose regular headquarters are in San Francisco. The decision to send Mr. Huttl to the Congress and on a circle trip of South American countries springs from our conviction that South American mineral resources are destined to play an influential part, not only in the defense of this Hemisphere, but also in the future mineral supply of the World. That being true, it becomes the duty of *E.&M.J.* to know at first hand the mining industry of Latin America and to acquaint its readers with prevailing conditions and potential developments in the southern continent. For nearly a year we have been doing this as well as possible with the friendly cooperation of contributors; but we now supplement that activity with direct editorial observation, the results of which we shall publish as promptly as possible.

Pan American Mining Congress

UNDER THE AUSPICES of the Chilean Institute of Mining Engineers, and with the official recognition of the Chilean Government, the First Pan American Congress of Mining Engineering and Geology will meet in Santiago, Chile, January 11–20, 1942. Here is a timely undertaking by our South American neighbors, and one greatly to be commended and supported by the mining fraternity of North America. In South America lies the great frontier of the mining industry in the Western Hemisphere, and the further development of its mineral resources will be a matter of international importance in the next decade.

That the United States looks approvingly on such development is seen in the active participation of personnel of the Bureau of Mines and the Geological Survey in exploration for and production of South American minerals in which the United States is deficient in the present emergency. But interest in South American mineral resources extends beyond the present because of their undoubted economic importance in the future of the World's mineral industry. For a quick appraisal of these resources two defense agencies in Washington—the Coordinator of Inter-American Affairs and the Economic Defense Board—are considering a proposal to send forty mining technologists into South America to supplement and expand the work already done by the Bureau of Mines and the Geological Survey.

The agenda of the Congress at Santiago provides expansive coverage of geology, mining, and ore treatment. Participating will be technologists from all countries in the Western Hemisphere. Gratifying attendance seems assured from Latin America, and a substantial delegation of engineers will be present from the United States and Canada. We hail the First Pan American Congress of Mining Engineering and Geology as the forerunner of such annual gatherings for many years to come.

Off Again, On Again

EMULATING the noble Duke of York, who marched his men up the hill one day only to march them down again, OPM recently issued and promptly withdrew an order curbing the use of lead foil for wrapping cigarettes and chewing gum. At least the order was suspended for thirty days, within which OPM will belatedly "get all the facts," and decide whether the order shall be revoked, modified, or put into effect unchanged.

Vigorous protest from Reynolds Metals Company turned the spotlight of publicity on the order. One sympathizes with Reynolds officials as they pondered the closing of two lead foil plants employing 2,000 workers. But one must be equally sympathetic with scores of leaders of other non-defense industries that find themselves out of business for lack of raw materials. They, like Reynolds, find it hard to believe the stories of metal shortage. But, unlike Reynolds, their plaintive voices have been ineffective in getting relief. Why the difference? One hesi-

tates to suggest that Reynolds is synonymous with big business, also engaged in making aluminum with government money, whereas the others are only small fry. Perish the thought that under the New Deal big business should be favored and little business thrown to the wolves. But OPM will have to be very convincing if it now decides to permit manufacturers to continue making so relatively needless a product as lead foil for wrapping gum and cigarettes, at the same time that it urges the lead mining industry to increase production for national defense.

Mexican Accord

OUR WASHINGTON CORRESPONDENT emphasizes this month the increasing dominance of political considerations in defense planning and international negotiations. The phenomenon is not new; it is merely becoming more evident. More than a year ago we began to point out that many acts of the Administration then and thereafter affecting metals and minerals could be rationalized only by giving undue weight to political expediency. Diplomacy was destined to supersede logic, because logic frequently leads to an impasse, and that was something to be avoided at all hazards in establishing the twin objectives of Hemisphere solidarity and Axis defeat.

The Mexican accord signed on November 19 is a case in point. A bargain was concluded in which the United States paid a high price for Mexico's active adherence to plans for Hemisphere defense. The agreement was as notable for what it excluded as for what it settled. Long outstanding property claims against Mexico are to be discharged by the payment of $40,000,000. On the other hand, no reference was made to United States interest in nearly a billion dollars of Mexican foreign obligations long in default. The United States agreed to resume buying silver direct from the Mexican Government to the extent of about six million ounces a month; and the Export-Import Bank is to loan money to expedite completion of a highway construction program in Mexico. But the agreement failed to resolve the oil expropriation issue, which has been the chief source of friction between the two countries. Probably the outstanding item of future economic importance was the agreement for monetary and financial cooperation in which our Treasury undertakes to stabilize the Mexican peso to the mutual benefit and advantage of the two countries.

The accord has little new significance for United States mining interests in Mexico, although the handling of the expropriation matter leaves something to be desired. Apparently the State Department weighed the interests of the oil companies, and its own earlier firm attitude in their behalf, against the desirability of international accord, and decided in favor of the latter even though that meant practically abandoning the oil companies to the mercy of negotiations on Mexico's terms.

Excepting the oil item, the accord should contribute to general improvement of our relations with Mexico, particularly through stabilization of the currency and the writing of a new trade agreement, which is still to be undertaken. With these things accomplished, with the war emergency out of the way, and with priorities no longer a handicap to export trade, Mexico should again become a good market for United States manufacturers, particularly of mining machinery. It looks as though Mexican mineral products are likely to find a market in the United States for some time to come, and the logical corollary would be a reciprocal market for mining equipment.

Germany's Mineral Supply

SPECULATION on the adequacy of Germany's supply of metals and minerals essential to the conduct of the war can be fairly dismissed after a study of statistics and facts outlined in an article published on succeeding pages. The validity of the information is unquestioned because it was gathered from official sources and through personal investigation by Charles Will Wright in his capacity as Foreign Mineral Specialist of the Bureau of Mines. If Americans have had any lingering doubts about Germany's capacity to wage war they will be disillusioned by even a casual glance at the figures.

Several conclusions seem inescapable. First, Germany began preparation for present hostilities in 1932, when a program of importation of strategic metals and minerals was started that expanded steadily each year until she felt prepared in 1939. Second, in marked contrast during the same period, our Congress ignored the warnings of the mining industry and postponed a program of building stockpiles of strategics until tremendous obstacles were interposed by the turn of events in 1939. Third, if this war is to be won, it will necessitate throwing the full weight of America's mineral resources into the balance with those of Britain, and producing machines of war in such number as to overwhelm the Axis powers. This will be no mean task, as Mr. Wright's figures show, but the program is now under way and will be carried to a successful conclusion.

Reflections from Washington

By Special Correspondent

MINERAL POLITICS is becoming more than ever dominant in defense planning and international negotiations. Goods control is just as important as gun manufacture in modern warfare. This was illustrated during November by numerous events. Japanese negotiations were really more importantly governed by mineral and trade considerations than by territorial desires. And Washington saw nothing more important in all hemisphere planning than the action of late November establishing military protection by Uncle Sam for the bauxite supply from Surinam.

No one who has followed the activities of United States uniformed forces could have been much surprised that there was a desire to establish a strong military unit in Dutch Guiana. However, the form in which the announcement was made by the White House reemphasized the fact that cooperation with Latin American neighbors is to be the basis of further military activity in the Caribbean and South American areas. Strengthening of our force to protect the flow of bauxite was important. But the establishment of a working base capable of supporting a large military operation if necessary was probably much more important.

That all of this was done with the active and direct cooperation of the Netherlands Government at London and of the Brazilian Government was also significant. This appears to be the first avowed step toward common military action with Latin American countries on land. It presumably is but one of numerous land base operations which have long been expected but not hitherto officially spotlighted. Wherever else the mining industry may be threatened with difficulty, it can expect Uncle Sam's soldiers to be furnished for protection of the lines of communication essential to strategic mineral supply. And this is not limited to the Western Hemisphere.

Copper Orders

PRODUCTION OF MORE COPPER in northern Michigan was definitely promised as the result of contracts finally signed during the early part of November. Comparable negotiations to stimulate even the high-cost producers are still being talked for other parts of the United States. No matter how much production may be so stimulated, the supply will be wholly inadequate to meet the demand.

Price problems have continued to bother Washington on all types of copper products. Pending the establishment of formal ceilings on prices, there have been repeated requests that manufacturers of various products, such as wire and cable, refrain from any further advances in their quotations. If there is not full compliance it is expected that a severe crack-down will be made; and Price Administrator Henderson may go so far as to suggest that copper be withheld from firms that allow prices to get out of line.

Tapering off of copper supply for users who will not get any of this metal after certain fixed dates has proved to be a difficult problem. Certain adjustments to prevent undue hardship on individual companies are being attempted. And, incidentally, OPA is watching sharply to prevent price advances on the products containing copper as a minor component. There have been numerous efforts through political channels to secure extension of time before complete shutdown of metal products plants.

Priorities Order P-56 Revised For Mining

MINING PRIORITIES are now consolidated under a special OPM Division, headed by Dr. Wilbur A. Nelson. Late in November Priorities Order 56 was revised to include repair and maintenance equipment and operating supplies and also new equipment for expansion. Important details of the order are given on page 97.

Machinery Prices

THE OFFICE of Price Administration has been conferring with important mine machinery and equipment people in an effort to prevent price rises that are not justified. They are still seeking some sort of price formula that will recognize a reasonable margin of profit without permitting wage and material advances to pile up percentage-wise in the payment which the mining enterprise must make.

Near the end of November conferences were held with various representatives of individual firms supplying this equipment. Then a general conference followed on Nov. 28 and 29. In this as in other machinery price control the OPA officials find that it is not practical to fix ceilings. Each job of design or fabrication is likely to be "special." There are, of course, many minor items, such as pumps, motors, etc., which are manufactured in quantities and for which definite price schedules have been in force in the past. On these, price control is not so difficult. But on big machinery jobs it is not practical to say "the prices must not advance higher than those which prevailed on ——."

Washington hopes that mere discussion and exchange of information between the government and equipment builders will suffice to keep matters well in hand. If in any case the government's own interest becomes seriously affected, specific action may be expected. Such a case might arise if Defense Plant Corp. or the Army were the purchaser of the equipment. But such occasions are relatively rare in the mining business.

Tax Plans

FANCIFUL FIGURES have been talked of in outlining tax plans for 1942. No one really expects the extreme estimates to be realized, but it is a serious mistake to ignore the general trend of this talk. The customary New Deal technic of getting legislation is being used. Extremes far beyond the objective sought are first described. Ultimately Congress and the public will accept proposals that will seem fairly mild by comparison with the extremes.

Withholding taxes would account for an additional $15,000,000,000 of income in 1942 if the extremists could be believed. Such Federal "take" would be accomplished by requiring everyone who disburses interest, dividends, commissions, fees, salaries, or other types of income to withhold for transmittal to the government 15 percent of these payments. It should not be assumed that this is a transactions tax because it would not apply to the gross on sales of goods, though it would apply to gross for service payments.

Some fantastic proposals have gone so far as to say that the person from whom this 15 percent was withheld should still be required to report it just as if it were income and subsequently pay income taxes on it. Such a tax on taxes is not likely, however.

Further Social Security taxes seem certain. The most important feature of the change in this regard is the fact that federalization of all unemployment as well as old age features is contemplated. It remains to be seen whether the various State delegations in Congress can be organized against the transfer of all this business to Washington control. But the percentage applied to payrolls next year, and probably the numbers of persons to whom Social Security taxes apply, will be substantially increased.

The first tax proposals would have given the Federal Treasury an additional $20,000,000,000. The argument is that at least that much additional money must be taken away from those gainfully employed to prevent serious inflation next year. It seems unlikely that so much will be made a tax, but the moderately radical planners think that a substantial part of the $20,000,000,000 could well be in the form of compulsory saving. Thus the Treasury would get the money for defense bonds from cool or unwilling lenders at low rates of interest. Some even propose that such borrowing be in the form of non-interest-bearing securities.

Domestic Manganese

IT IS OFFICIALLY ANNOUNCED that Metals Reserve Co. will "now consider contracts with producers for low-grade manganese and chrome ores with a minimum content, respectively, of 40 percent manganese and 40 percent chrome." Domestic producers have claimed that if reasonable specifications were set their production would substantially aid in the defense program. Large actual realization of production in tonnage is not, however, expected in Washington.

Imports of the desired grade of manganese ore continue at a reasonable rate. The stockpile situation appears comfortable. This is not causing any relaxation of the effort to build more reserves. But OPM officials clearly indicate that they would be very happy if the copper, lead, and zinc situation were as easily handled as are manganese and chromium.

Further development continues on new processes for recovery of manganese in secondary products of desired grade by leaching and special processing methods. Most of this work is continuing under Bureau of Mines direction. Certain commercial efforts are also being talked of, though no contracts had yet been announced officially for such undertakings up to the first of December.

News "Fines"

Industrial Silver Up—The agreement between Mexico and the United States by which the Treasury will acquire newly mined Mexican silver at the Treasury's buying rate resulted in an unexpected advance in the price of foreign metal for general industrial consumption. To divert silver from the Government's growing "stockpile," the quotation to the trade was raised on Nov. 28 to 35¼c., New York. This marks the first change in the open quotation since June 17, 1940.

Lead and Zinc—Price questions abound in Washington with respect to all nonferrous metals. Repeatedly it has been rumored that the price of lead would be further advanced; and with equal frequency Price Administrator Henderson denies this. But as December opened it was confidently expected that some advances will be made to encourage high-cost production. The uncertainty at that date appeared to be whether a general price advance would be permitted, or whether (more probable) some special contracts comparable with those for copper would be made with individual high-cost producers.

Over-Price Zinc—Official conclusions regarding charges for zinc above price ceilings were published during November by OPA. That agency describes evidence gathered regarding sales at prices of 16 to 17c. per pound and at other figures above the agreed prices. The parties charged with violation are Hegeler Zinc Co., of Danville, Ill.; Sol M. Marks, of Cleveland, Ohio, agent and dealer for the Hegeler Co.; Tyroler Metals, Inc., Cleveland; and Adam Hope Co., Detroit, dealers who handled Hegeler zinc. Mr. Henderson has suggested that "buyers of the zinc might well explore the possibilities of proceedings to recover the over-payment."

Gypsum Acquittal—The five firms and eight individuals indicted for price fixing of gypsum products sold to the building industry were acquitted by a directed verdict of the District Court at Washington, D. C., on Nov. 19. The presiding judge held that the government had failed to produce convincing evidence of conspiracy and restraint of trade which was charged in the indictment.

How Many Shifts?—Industrial leaders are being compelled to come to Washington to explain why their establishments are not working more shifts per week. Any mining or metallurgical enterprise producing a strategic material may be called on any time to tell why it is not working at least the equivalent of 15 or 18 eight-hour shifts weekly. The government is getting hard-boiled on this question and is likely to crack down with orders if "requests" are not adequate to get results. Lead and zinc producers have been spotlighted by criticism already.

Overtime Rule—Calculation of overtime for employees who get pay for certain periods when they do not work has raised an important question at the Wage-Hour Division. It is now pointed out that the requirement of time-and-a-half for periods worked beyond 40 hours per week applies to actual work time calculation. An employee who is on the job by the employer's direction but not actively engaged on his duties is considered to be working during this time. But if he is being paid during vacation, illness, or other absence from his duty station or place of employment, the time for which he is so paid does not count as work time when figuring when the time-and-a-half rate begins to apply.

Metal Packages Restricted—Use of wrapping materials is being severely restricted by OPM rules. Many container regulations have been released, including those which forbid use of metal containers for distributing petroleum products. Of still greater interest to the metal industries was the limitation order of Nov. 24 forbidding the use of tin foil or lead foil for wrapping tobacco, chewing gum, beverages, confections, and a number of other products. This action was strongly opposed, and late in November OPM suspended the effective date of the order for 30 days with implication that the order might be modified or possibly not invoked at the end of the 30-day period. Extreme and rigid rulings of this sort are being imposed with increasing frequency.

Explosives Licenses—The bill providing control of explosives by Federal license passed the Senate Nov. 17 in a form somewhat different than the House bill. But it appears probable that during December this measure will go to the President and receive his approval. When enacted it will provide that the Bureau of Mines establish a licensing system for those who produce, distribute, or use explosives. This will extend clear through to the licensing of mining employees who require explosives for their regular use. The nominal objective is prevention of sabotage by misuse of explosives.

Copper Probe—Congress has heard so much complaint about copper supply that it undertook an executive-session investigation on Nov. 28 through a House appropriations subcommittee. A similar inquiry was scheduled to start Dec. 9 in the Senate special committee headed by Senator Truman. Two questions are being asked: Why do we not produce more copper? Why are our constituents not given as much copper as they want? Spokesmen of the mining industry, as well as OPM officials, are telling the story of their difficulties.

Titanium Control—Titanium dioxide has been placed under priority control with the requirement that 20 percent of production must be reserved by each manufacturer to fill defense orders under government allocation. Shortage of imported titanium raw materials is blamed for current difficulties. But it is explained in Washington that the measure fixed by OPM, Order M-44, was more precautionary than corrective, as actual serious shortages of titanium pigment have not been experienced as yet.

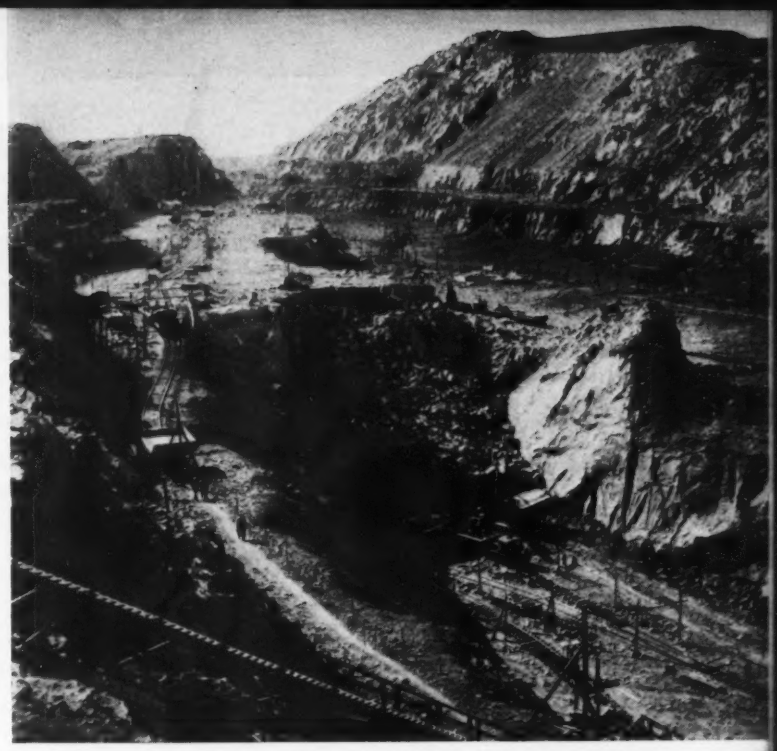
Imitate Europe—To secure economy in use of materials, especially metals, the bureau of industrial conservation in OPM is seeking to get re-design of many machine and equipment parts. It is argued that the United States must now adopt the economic philosophy of Europe, where the relative importance of materials used and labor needed for manufacture has always been different than here. Even when more labor is needed to make equipment which uses less metal, it may be necessary to adopt that policy for the present in the United States. Hitherto the use of more metal to save fabricating labor was considered better economics. But all matters of economy go by the board when material shortages of present magnitude have to be solved.

Non-Defense Mining—Fear was expressed quietly during October and November that OPM would rule silver mining and gold mining to be non-defense activities. This fear was lessened by the formal extension of Preference Rating Order P-22 to include gold placer mining, as well as numerous other mineral developments in the fields of natural gas, petroleum, and other products. Mining priorities have also been extended to operations in the Philippine Islands.

Western Phosphates—Secretary of Interior Ickes is making another effort to develop a phosphate products industry in the Northwest. He has named a new committee to survey possibilities and recommend location, process, and market objectives. Obviously the unnamed purpose is to establish in the Columbia River Valley a fertilizer program comparable with the T.V.A. project, using Federal power and Western phosphate as raw material.

Price-Control Bill—As December opened, the situation on price control legislation was confused badly. Congress is proving reluctant to grant any one of Henderson's temperament such sweeping powers as would be given under the most conservative bill which Congress thinks would have any value whatever. It appeared at that date that no enactment of any sort would be practical until February.

Germany's MINERAL Supply



Above the Arctic Circle lies Sweden's famous iron ore deposits at Kirunavaara, estimated to contain around 1,500,000,000 metric tons of high-grade iron ore. Swedish iron ore comprised 40 percent of all German imports of iron ore in 1938

A statistical study of resources derived from imports before hostilities in 1939, and from countries occupied or controlled since the war began

IT IS AXIOMATIC that metals and minerals play a primary rôle in modern warfare, and that the nation with the greatest mineral wealth has an advantage over less fortunate peoples—provided the mineral wealth is promptly converted into fighting machines and munitions. Mere possession of mineral resources is of little military value unless they are used for combat quickly and in overwhelming quantity. So despite the fact that the British Empire and the United States have controlled the greater portion of the world's mineral wealth, Germany has thus far made better military use of her mineral supplies and has thus far easily conquered her adversaries.

That Germany began to plan her conquests in 1932 is suggested by the record of her imports of certain strategic minerals as shown in the accompanying table, which is compiled from German Government statistics. Between 1932 and 1939 annual imports of copper ore increased three times; of manganese ore, four times; of nickel metal, seven times; and from 10 to 25 times for tungsten, chromium, molybdenum, titanium, and vanadium ores.

Current mineral supply is set forth in the following information obtained by Charles Will Wright, Foreign Mineral Specialist, U. S. Bureau of Mines:

Iron Ore—In 1940, 46 percent of the world's iron ore output was produced in the countries now controlled by Germany. Germany has recently occupied the iron mines of Krivoyrog, in the Ukraine, said to contain a billion tons of 57 percent iron ore and to have an annual production of about 18,000,000 tons. This new source of supply, together with the high-grade ores from Sweden, places Germany in a better position to supply the requirements of her steel plants than that of Great Britain and the United States.

Pig Iron—In German occupied areas 44 percent of the world's output of pig iron was made in 1940. It is probable that the blast furnaces in the Ukraine were destroyed and it may take a year to rebuild them. But regardless of this additional capacity Germany is being well supplied with all of her pig-iron requirements from the blast furnaces within her borders and the adjoining areas in France and Belgium.

Manganese Ore—Germany's shortage of manganese ore was becoming serious, but by the occupation of the Nikopol manganese mines, in the Ukraine, Germany has acquired an almost inexhaustible source of supply, with an annual output of 1,200,000 tons, or nearly double that of the Western Hemisphere. The areas now occupied by Germany produced 27.5 percent of the world's manganese ore output in 1940.

Chromite—Within the German occupied countries the production of chromite in 1940 was 10 percent of the world's output. Although it was less than the German imports in recent years, this together with stocks on hand is probably sufficient for all necessary requirements during the next few years. Both Great Britain and the United States depend almost entirely on overseas imports from Africa, India, New Caledonia, and the Philippine Islands for their chrome requirements.

Tungsten—This is one of the few mineral products not produced in sufficient quantity within the new German controlled area. Germany's nearest source of supply is in northern Portugal, which district in 1940 produced 4,858 tons of concentrate, about one-half of which is said to have been shipped to Germany. Germany's requirements are over twice the amount now obtainable, but imports were ex-

ceptionally large during the years preceding the war, and in 1938 amounted to over half of the world's production.

Nickel—This is another metal produced only in small quantities in the German-controlled countries, Norway and Greece being the principal producers. Although a heavy importer in past years, Germany's present supply is probably small. However, if she has access to develop the new nickel mines at Petsamo, in Finland, this will be her most important source of supply. In 1940 only 2.5 percent of the world's output was produced in German-occupied countries.

Copper—Germany's annual copper consumption has reached 300,000 tons in recent years, judging from imports and production. With control of the Bor mines, in Yugoslavia, the exports from Spain and those from Norway and Sweden, Germany's position has been greatly improved. Large stocks of copper are said to have been acquired in France, Belgium and Holland, and estimates indicate that Germany will have a sufficient supply for the next few years. Every effort is being made to replace the use of copper by plastics, aluminum, and other metals. In 1940 about 7 percent of the world's output of copper was from German-controlled countries.

Lead and Zinc—The occupation of the lead-zinc districts at Katowitz, Poland, and at Trepca, Yugoslavia, has added an annual production of about 175,000 tons of zinc and 32,000 tons of lead under German control. Italy is also a large contributor, and Tunisia and Algeria produce annually over 20,000 tons of each of these metals. Thus both in lead and zinc Germany controls a sufficient annual output to meet her requirements. In 1940 German-controlled countries produce 33 percent of the world's zinc output and 25 percent of the lead output.

Antimony—In Europe antimony is produced for the most part in Yugoslavia, which accounts for 10 percent of the world's output. This new source of supply, together with another 10 percent from the rest of Europe, will probably give Germany a sufficient tonnage of antimony to meet her needs.

Tin—There are no important tin mines in Europe, and Germany depends entirely on imports from the Far East and Bolivia for her requirements. Germany is believed to have had large stocks in 1938 and to have acquired additional stocks in Holland. Germany has developed the BN and other bearing metals for general use which do not contain tin, which is used only in the high-speed bearings. Tin for the canning industry has been largely replaced by substitutes.

Bauxite—The extensive bauxite deposits in France, Italy, Yugoslavia, and Hungary are situated in a much more favorable position for transport to the aluminum reduction plants than the deposits from which the aluminum plants in the United States and Canada get their supply (principally British and Dutch Guiana). Present output in Europe, which is 50 percent of the world's production, is considerably more than that in the Western Hemisphere, or 38 percent, and this output could easily be doubled.

Aluminum—In 1933 the United States output of aluminum was more than double that of Germany, and in 1940 Germany produced 240,000 tons, against 187,000 tons in the United States. Aluminum production in German-controlled countries, France, Italy, Norway, and Switzerland, in 1940 was 15 percent greater than the combined output of the United States, Canada, and Great Britain, and amounted to 47.5 percent of the world's output. A recent survey of competent experts shows that aluminum plants in German-controlled Europe will have a total capacity of 560,000 tons by the end of 1941. New plants under construction and consideration in the United States, Canada, and Great Britain when completed, probably in 1943, will have a total capacity of over 900,000 tons.

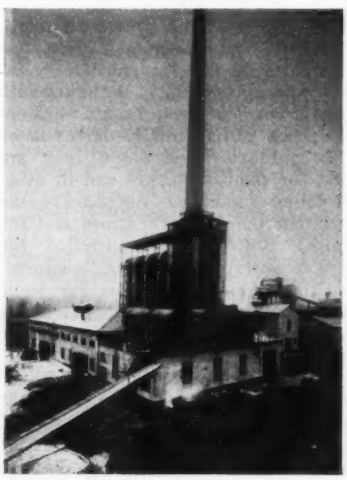
Magnesium—No official production figures are published in Germany on magnesium output, and estimates vary from 18,000 to 60,000 tons, the most recent being 40,000 tons for the German-occupied areas. In 1940 the combined British-American output was only 12,000 tons. Large plants are under construction in the United States and by 1942 domestic production of this light metal should be about 180,000 tons.

Mercury—As Italy is the largest producer of mercury, followed by Spain, the two countries controlling 75 percent of the world's output, it is not a question of Germany's requirements but one for the Allied nations to worry about. In recent years new mines have been developed in the United States and Mexico and their output is about sufficient for United States needs. Production from these mines in 1940 was 1,704 metric tons.

German Imports of Certain Strategic Minerals

	Metric Tons							6 mos.
	1932	1933	1934	1935	1936	1937	1938	1939
Tungsten ore	1,615	3,614	4,385	7,681	8,726	11,372	14,200	5,731
Tungsten metal and scrap (incl. chromium and cadmium)	630	1,317	2,877	1,611	4,114	6,605	7,590	4,930
Nickel ore	17,694	34,513	37,609	29,013	17,651	19,990	34,215	15,800
Nickel metal	565	1,968	4,060	5,929	3,392	3,365	3,984	2,164
Chromium ore	42,128	47,370	76,983	95,440	123,375	132,162	176,406	110,131
Copper ore	237,878	240,902	324,921	400,538	482,471	555,577	653,931	336,523
Copper metal	138,100	154,600	182,200	153,365	127,549	169,920	272,400	89,355
Copper scrap	17,527	17,911	14,039	19,891	24,372	32,703	27,308	12,107
Manganese ore	107,000	132,000	223,000	394,256	229,634	554,170	425,785	153,757
Molybdenum, Uranium, Titanium, Vanadium ores	2,848	8,945	20,595	27,359	24,512	41,861	64,179	21,365

Modern American equipment is used at Kiirunavaara



Germany is reported obtaining more copper from the Imatra smelter that treats concentrates from the Outokumpo mine, in Finland

The mine and mill of the Potash Company of America are 22 miles east of Carlsbad by rail

Potash Company of America—Producer of Sylvite at Carlsbad

An account of its development, mining, and treatment operations

R. A. Pierce
Assistant General Manager

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Consulting Engineer

DISCOVERY OF POTASH in commercial quantity in southeastern New Mexico, and its development, successfully culminated the efforts started over 25 years ago by the Federal Government to free this country from dependency on a foreign source of supply.

Up to the time of the first World War, Germany, as the world's largest producer of potash, dominated the industry, making it necessary for American agriculture to depend entirely on imports controlled by a foreign monopoly. With that war came an embargo and transportation difficulties which curtailed imports so seriously that it was not long before agriculture, which accounts for about 90 percent of American potash consumption, was deprived by scarcity and price of this most essential form of plant food. An effort to avert what was fast becoming a potash famine was inaugurated by private industry, by the erection of plants at different localities throughout the United States, attempting the extraction of potash from hitherto unexploited sources such as vegetable and animal material, cement dust, the deposits of dry lake beds, brine deposits, and many others. That partial relief could be obtained from such sources was demonstrated when toward the end of the war plants with a capacity equal to half of the normal requirements of that time were in operation.

With the close of the war and the establishment of normal relations with Germany came the return of potash imports, and with it the restoration of pre-war potash prices. This was competition that few of the plants could endure, and of the war-time sources only one company, the American Potash & Chemical Co., operating on brines of Searles Lake, in California, has been successful in developing into a producer of any magnitude.

Today, with the nations of the Old World again in the throes of war and the discontinuance of imports because of lack of transportation, we find instead of potash history repeating itself, that there is a self-sufficient American industry already in active operation, assured by the exploration and development of the New Mexico potash deposits and the brine of Searles Lake.

The New Mexico potash deposits from which this nation is now receiving the major portion of its potash salts are being developed by three companies—namely, United States Potash Co., which came into operation in 1931; the Potash Company of America, which began production in 1934, and Union Potash & Chemical Co., which started operations in 1940. That these companies can meet foreign competition on potash salts is answered by the fact that during the last seven years two of them, with the aid of the American Potash & Chemical Co. and other smaller plants, have not only succeeded in marketing over half of the domestic needs, but have released it to the American farmer at a price substantially lower than the previous prices. Now with the third company in full operation on the New Mexico deposits and the increased production being secured through expansion of plants and mines of the other companies, the potash industry has reached the point where it can supply all of the normal domestic requirements of this essential commodity.

The deposits from which the New Mexico companies are mining and refining potash salts are in the upper salt series of the Delaware Basin, a subsidiary of the Permian Basin which covers southeastern New Mexico and southwestern Texas, and which was formed during the final stages of the Permian salt-building period.

The lower salt series, or Castile Anhydrite, of the Delaware Basin, which is barren of potash salts, is composed mainly of anhydrite, built up in layers with occasional beds of halite. Geological evidence indicates that a marginal limestone reef was formed concurrently with the depositional filling of the basin, which, along with the anhydrite deposits, attained a maximum thickness of 2,000 ft. prior to the re-establishment of a connection with the ocean to the south. During this period the deposition was changed from anhydrite to halite and there was formed what is known as the Salado Halite, or upper salt series. Potassium salts which occur only in the Salado Halite were first recognized from brines and latterly polyhalite chippings secured from oil-well tests drilled through the halite measures. These findings were followed by both public and private core drilling that resulted in the discovery of the first commercial deposits of soluble potash salts.

In the Salado Halite, or upper salt series, which constitutes the large accumulation of saline deposits that lie between the Castile Anhydrite and the

Rustler sediments over the Delaware Basin, are found interspersed throughout its thickness many beds and masses of potassium salts.

The U. S. Geological Survey has, through information derived from salt cores, secured in the exploration of the Salado Halite, recognized, classified, and correlated as beds 40 horizons that show enrichment of potash. The major part of these beds are polyhalite which appear at varying intervals throughout the 1,500 ft. of depositional halite. Due to their continuity, these beds have been recognized in a majority of the test wells drilled within the confine of the Delaware Basin, and by correlation they have been used most successfully as a guide in the search for sylvite, as they represent what might be called "markers" outlining the depositional cycles which were repeated in periodical sequence throughout the salt-building era.

The areas in which sylvite and associated potash salts, such as carnallite and langbeinite, are found occupy laterally only a small part of the Delaware Basin and occur in the upper portion of the Salado Halite with mineralization restricted to a vertical section which averages 250 ft. in thickness and which is designated the "Sylvite Zone." In this zone the halite is divided into intervals by thin beds of polyhalite and into two sections by a very pronounced and persistent bed of anhydrite that is usually capped by polyhalite and used in drilling as the key bed or marker to establish the identity of the sylvite zone. Each of these intervals represents a depositional cycle or successive period in which sylvite was deposited, and the presence or extent of mineralization of the different intervals has been established and correlated by examination and analysis of cores secured in test core-drilling.

From the information thus obtained it was learned that in five intervals the concentration of sylvite had been high enough to be considered a bed and that in general the lateral extent or area covered by each successive bed cut was smaller but richer. Three of these beds lie above the anhydrite, but none has attracted attention, owing to lack of sufficient concentration to be of commercial interest. The lower sylvite section lies immediately below the anhydrite and contains two beds. This section varies in thickness from 50 to 90 ft. and is divided by polyhalite into two intervals. In the upper interval the bed usually occurs in one or more rather thin rich sections that are separated by halite containing scattered blebs of sylvite.

No sylvite at present is being extracted from this interval. Langbeinite in appreciable quantity has been found in certain localities in it, and extraction is being made at one place through the shaft of the Union Potash company. In the lower interval three areas have been discovered in which the sylvite deposit is large enough and rich enough to warrant development. To each a shaft has been sunk and extraction is being conducted on a commercial scale by the three independent companies mentioned.

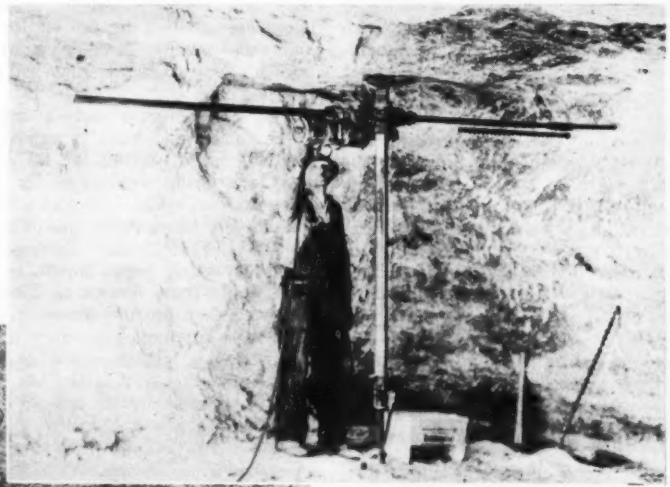
The bed designated as No. 4 was deposited in what appears to have been the lagoonal area of separate basins, as the limits of each of these areas have been well defined and no connection of this or other beds below the anhydrite is evident. In the upper section where the beds cover progressively larger areas it may be possible that some are continuous. The deposits are irregular in outline and lenticular in form, but vary somewhat in average thickness and richness. The thinner deposits are the richer, but all of them appear comparable in the average amount of KCl contained.

The first discovery of sylvite dates back to 1925 by the finding and identification of sylvite crystals from cuttings obtained by the Snowden-McSweeney Co. in drilling an oil test. Following this important discovery an exploration company to prospect by core drilling the area adjacent to this discovery well was formed. On this work the company spent several years before locating what was later recognized as the first commercial body of soluble potash to be found and operated on this continent. To prove and develop the area established by drilling, a 1,000-ft. shaft was necessary. This was started in 1929 and completed early in 1931, when the No. 4 sylvite bed was cut. The operating company which was formed to develop the property was the United States Potash Co.

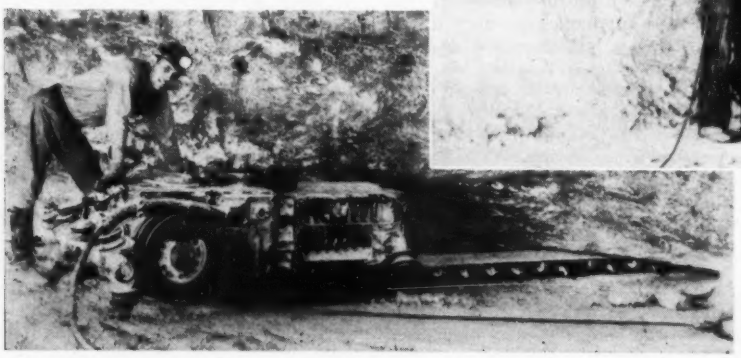
The second deposit of sylvite in commercial quantity to be discovered was located by Potash Company of America on acreage selected after study had been made by engineers and geologists from data secured and released by the U. S. Geological Survey on core tests made by it in the Permian Basin. The first selections were made in the early part of 1931 and comprised both Federal and State land. Exploration work by core drilling was started shortly thereafter.

For prospecting, the company contracted the services of two drilling companies, both of which had just been released from Government tests and were familiar with the coring of highly soluble potash salts. Locations for testing, depths to be drilled, character of drilling solution, receiving and examination, and analyses of cores were all handled by the company's staff.

In drilling the sedimentary red beds or "Rustler" formation that covers the Salado Halite the customary pro-



Drilling is done with post-mounted electric augers equipped with high-speed detachable bits



Undercutting the sylvite. A 9-ft. bar is used

cedure is to use cable tools until the halite is reached, as it is much cheaper than core drilling, and the information desired, such as water, horizons, and character of rock, can be determined with sufficient accuracy thus.

After landing on the halite bed with not less than 5¾-in. casing, and before core drilling is started, a seal is made between the casing and the hole. The seal is made to eliminate the possibility of water entering the hole while core-drilling in the halite and to prevent the loss of drilling solution through the overburden during the process of forcing the salt cuttings to the surface, where they are settled out before the brine is recirculated. The drilling solution used at first was a saturated solution of sodium chloride, magnesium chloride, and potassium chloride. Later, when it became available, carnallite or sylvinite was substituted for the salts. With a saturated solution no leaching appears on the cores and an accurate determination of the mineralization can be made.

A double-core barrel 16 ft. long is used. The size of core cut is 2½ in. in diameter and runs of 15 ft. can usually be made before lifting. The cutting bit is slightly different from the one ordinarily used where the drilling fluid passes over the core and inner portion of the bit before returning across the face. With a salt bit the walls are double and the solution, instead of washing the core, is forced out through the face of the bit, which eliminates all contact between the solution and the core while cutting. Erosion is also eliminated thus. Diamonds are used for cutting and in most cases the recovery of full-sized cores closely approaches 100 percent.

Nearly two years, or until January, 1933, were spent by Potash Company of America on the drilling and completion of 15 tests that discovered and outlined on the No. 4 bed the second commercial deposit of high-grade potash (sylvinite). Before making final selection of all the ground now held under lease, five more core tests were finished. In all, 31 have been completed.

To develop and operate the lease holdings of Potash Company of America required not only the sinking of two shafts but the construction of a railroad spur 22 miles long to connect it with the Santa Fe Railway at Carlsbad. The spur, which is owned and operated by the Santa Fe, was constructed during the time that Potash Company of America was sinking its No. 1 shaft. Both were completed early in January, 1934, and the first car of unrefined potash salts to be shipped left the mine the following month.

In selecting a surface location for the shaft, the company was fortunate in finding one that not only satisfied the surface requirements needed for milling, refining, and railroad yards, but also those for underground transportation as well, as it cut the deposit in its deepest part, thereby insuring favorable grades for most haulageways between the working faces and the shaft bottom.

A track-mounted loader in action. The box-type scraper, when filled, holds 1¼ tons of potash

The No. 1 shaft, or hoisting shaft, was sunk to a depth of 1,086 ft. and contains three compartments—two for hoisting in which the cages and underslung skips operate in balance, and one for pipes, cables, and ladderway.

To reach the deposit, which lies at this location 994 ft. below the surface, it was necessary to sink through 430 ft. of overburden (the Rustler Red Beds) before reaching the top of the halite. This formation—the Rustler—is, from the top down, made up of gypsum, red shale and sand, anhydrite, limestone, and red sandy shale. With the exception of limestone and anhydrite, the other sections of the formation need heavy and close timbering and where water is present require concrete lining. In the limestone, which averages about 30 ft. in thickness, was found the heaviest flow of water. In the No. 1 shaft a flow of 350 g.p.m. was encountered, and in No. 2 shaft, which is 700 ft. away, the flow increased to over 1,000 g.p.m. The water is confined to a 10-ft. section that is very porous and honeycombed. The static head is 190 ft. and when free to flow diminishes in quantity but slightly. Water for industrial purposes is secured from this source. It is high in mineral content, clear, and has a temperature of 69 deg. F. All water is sealed away from the shafts by concrete linings and piped to large sumps cut in the anhyrite, where it is stored for pumping to surface as required. Supply and pumping are controlled automatically.

Sinking through the salt bed offered no serious difficulties, as the vertical side walls are self-sustaining and need only enough timber and lagging to protect the salt from moisture and form the compartments through which fast-moving skips can be safely guided.

The shaft was sunk 92 ft. below the potash bed to secure a gravity flow between the car-dumping station and the skip-loading station. To load the skips for hoisting, which is handled automatically, the potash is delivered by mine cars in trains to the shaft station. Cars are fed by gravity to the rotary dump for unloading onto an apron feeder. The feeder, which will handle 240 tons per hour, conveys it to a single-roll toothed crusher for reduction to a maximum 5-in. size. From the crusher it drops into a 100-ton pocket, to be drawn and measured automatically into the skips by a double rotary skip loader located 80 ft. below the mine level. The loader is operated by the skips as they come into and leave the loading position.

Plans for the development of the deposit were formulated after the results obtained from core drilling had been interpreted and the physical characteristics of the potash deposit and overlying strata determined.

The potash deposit, which is in the form of a bed, is defined as a section of the immense halite beds in which halite and sylvite form the integral parts. This intergrowth of massed crystals can usually be distinguished from the barren halite by the pinkish color which is imparted to the mass from the presence in minute quantity of ferric oxide, which coats, in the form of a thin film, most of the sylvite crystals. Sylvite (potassium chloride) in its natural state is colorless, or white, waxy in texture, and highly soluble in water. In potassium, the element of value, it is the highest of all potash-bearing ores, and thus the most desirable. The equivalent, potassium oxide, expressed by the symbol K_2O, is the standard by which commercial comparisons are made and the richness of the bed or purity of the product is based.

Dissemination of the sylvite crystals is not uniform throughout either

the sections of the bed or laterally. In section the bed varies in value between the bottom and top, with the richer portion in the upper half, where it is not unusual to find lenses of pure sylvite 8 to 10 in. in thickness. Laterally the values seldom remain uniform for any great distance or area.

In forming, the bed, being of depositional origin, was laid down in more or less of a horizontal position, over and in conformity with the depressional area at the top of what was then a bedding of halite about 500 ft. thick. After completion of the sylvite beds, saline deposition continued, and several hundred feet of salt was formed before the Guadalupe uplift occurred and the salt bed was elevated and eroded, leaving on an average 600 ft. of salt cover over the potash deposit. During the time of re-elevation, major folding occurred, which placed the greater part of the potash bed in the form of a broad synclinal basin. The minor folds, rolls, and other irregularities encountered in development are attributed to the unevenness of the original sea floor at the time of deposition.

For extraction of the potash bed the panel system is used, with the haulage entries which connect the production panels with the shaft bottom laid out at right angles to one another and driven on courses which cross the synclinal axis of the bed at a slight angle.

Three entries 60 ft. apart from center to center are used on these main development headings. The two outside entries driven 24 ft. wide are used for return airways. The center entry, 15 ft. wide and 8 ft. 6 in. high, is used for haulage and intake air. These entries, while being advanced, follow the floor of the bed. They are extended for about 1,000 ft. before the center entry is stopped to permit irregularities of roof and floor to be removed by grading. Heavy steel rails, bonding, and trolley lines are then placed and the entry is ready for further extension. Fine salt, from undercutting, is used for ballast under the wood ties, and once it sets the alignment and grade of track are maintained without further attention.

As a protection to all main entries, a barrier pillar 250 ft. wide is left on each.

Panel entries are turned at right angles off the main entries at 900-ft. intervals and advance 1,500 ft. before meeting the next haulage entry, which parallels the main entry. By this system the mine is divided into production panels 900 ft. wide and 1,500 ft. long for extraction by the room-and-pillar method. These panels have been laid out on the advance-retreat system so that pillars may be pulled from any part at any time at option. Due to the roof's strength and its resistance to shear, the high resistance of the pillars to crushing, freedom from bottom movement or heaving, and the non-occurrences of falls or caves over the unsupported rooms and breakthroughs, no timber is used. The pillars will stand indefinitely without loss or deterioration and are left temporarily in reserve to favor production from development.

Rooms are turned on each side of a three-entry heading and spaced 90 ft. apart; they are driven 30 ft. wide and 400 ft. long with a 30-ft. wide crosscut turned off every 60 ft. This system, by which 66⅔ percent of the ore is extracted on first mining, is used throughout the mine where scraper loading is being done. In panels where track loaders are used the percent of extraction is the same, but instead of a face 30 ft. wide in rooms and crosscuts, the width is reduced to 22 ft. to accommodate the reach of the loader gathering head.

The Operating Cycle

In advance of all faces the operating cycle is comparable and includes first the making of a cut across the face at the bottom of the potash bed; the cut, 6 in. high and 8 ft. 6 in. deep, is made by a short-wall undercutting machine having a 9-ft. cutter bar and a 50-hp. motor. In operation the machine advances across the face at a maximum feed of 5 in. per minute. The bits are alloy steel, and a set will cut 150 linear feet on an average before being replaced by resharpened bits. Reforging is done by a forging machine on the surface. The purpose of undercutting is (1) to assist in blasting down the face by providing a second free face to break to, thus reducing materially the amount of powder required; and (2) it increases the amount of potash that can be pulled per round. The increase over solid shooting averages about 100 percent, though this depends somewhat on the length of cutter bar used.

After the cutting is completed, the place is drilled for blasting by post-mounted electric auger drills. To blast a 30-ft. face, 30 to 42 holes are necessary. They are drilled in vertical rows of three or four, depending somewhat on the height, with the rows spaced 40 in. apart horizontally. The augers are equipped with high-speed detachable bits which cut a 1⅝-in. hole to the depth of the undercut at an average speed of 28 in. per minute.

Blasting is done with Hercomite powder loaded by the shot-firing crew. Electric delay caps are used to give proper rotation of firing, as all faces of a block or panel are fired at once by closing the final blasting switch on the main entry. A powder magazine large enough for two days' supply of powder is provided underground. Two specially built powder cars are used to distribute powder to the faces. As a rule, each place blasted requires about eight hours to clear before loading is started. To load the broken potash into cars for transportation to the shaft, two types of mechanical loading machines are used—scraper loaders and track-mounted conveyor-type loaders.

For scraper loading the width of room face is 30 ft. The place is double-tracked on 10-ft. centers and rails are kept, on an average, about 60 ft. from the face. Crosscuts which are driven the same width as the room are loaded out by turning the loader crosswise on the near-side track. This places the delivery boom in position to load the cars in train on the far track. The loader, which is track-mounted, is equipped with a three-drum hoist operated by a 40-hp. electric motor. The scraper is of the box type and drags, when well filled, 1¼ tons of potash.

For the track-mounted conveyor-type loader, the face width is 22 ft. and one track is used which is advanced every cut to within 9 ft. of the face. The loader is rated at 5 tons per minute and has a 13-ft. gathering head that moves horizontally and vertically by hydraulic control. The digging and gathering onto a centrally located conveyor is accomplished by arms that are attached at intervals on an endless chain and mounted on each side of the conveyor. The loader is serviced with 5-ton cars by a battery locomotive.

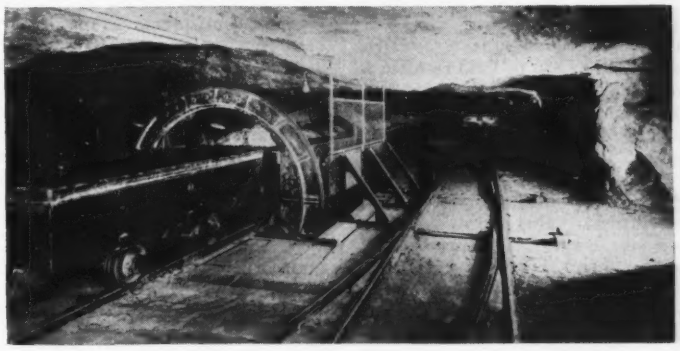
All of the production is transported from face to shaft bottom in 100-cu. ft. cars. The body of these cars is built of copper-bearing steel and is of the rectangular box type, equipped with spring-mounted couplings and swivel links. The trucks are equipped with 14-in. Timken-bearing wheels and four wheel brakes. For haulage, four types of locomotives are used—straight trolley, trolley with cable reel, combination trolley and storage battery, and straight storage battery. The latter type, which are 6-ton locomotives, are used primarily at the face to serve the loading machines where hauls are short and transfers rapid. The combination locomotives are used to furnish empties to and gather loads from the service locomotives. They operate between the rooms and the main-line partings where trips of 20 cars each are formed for transportation to the shaft bottom by the large and fast trolley locomotives. Rails of 56-lb. weight are used on all main lines and panels.

A 200-kw. motor-generator set is used to furnish d.c. current at 250 v. for trolley operation. Charging stations equipped with cranes and small motor-generator sets transfer and recharge the storage batteries. The combination locomotives are charged direct from the trolley lines either while at rest or when running on the trolley.

Ventilation is maintained throughout the mine by a 5x7-ft. pressure fan, powered by a 125-hp. synchronous

Likewise used for loading is this Jeffrey track-mounted conveyor-type machine, of 5 tons-per-minute rated capacity

A loaded train en route to the shaft. The trolley locomotive is one of four types used underground



The train is dumped a car at a time into the pocket at the shaft through which the sylvite is hoisted to surface

motor, and capable of producing 150,000 c.f.m. at a water gage of 3½ in. The fan is on the surface at the No. 2 shaft and operates normally as an exhauster, but can quickly be changed by means of four control doors into a blower, if its use in that capacity becomes necessary.

Up to the present no explosive gases have been met, but the same precautions were taken in laying out the ventilation that are used in a mine in which gas is being liberated. No open lights or smoking are allowed. Intake air is delivered through haulageways and each panel is on an independent split. Concrete overcasts are used on each panel and salt stoppings are built in each break-through.

Power is carried into the mine at 2,300 v., a.c., by a cable suspended in No. 1 shaft. Distribution lines carry it to the different areas, where by transformers in each panel it is dropped to 220 v. for use on all portable equipment.

Hoisting is done by a double cylindrical-drum hoist geared to a 600-hp., d.c. motor which receives its power from a 500-kw. motor-generator flywheel set. The drums are 5x9 ft., the rope is 1¼ in., and the rope speed 1,250 f.p.m. The headframe is steel, 126 ft. high, and the skip travel is 1,125 ft. between stations. The skips dump into steel bins with a storage capacity of 1,300 tons. From the bins the potash is conveyed to the mill for crushing.

The company operates its own power plant for generation of the electric power used in all operations connected with mining and refining of potash. Power is produced by seven diesel engines with direct-connected generators and exciters.

Engines having a total of 8,000 hp. vary in size from 400 to 2,400 hp. Three of the engines with a total rating of 1,750 hp. are of the four-cycle type using oil as fuel; the other four, totaling 6,250 hp., are of the two-cycle, air-injection, trunk-piston type, using natural gas as a fuel, but equipped so that they can be converted to oil in a few hours' time. Selection of fuel is governed by the economy effected when calculated on a B.t.u. cost basis.

Total generator capacity is 6,650 kva. at 80 percent power factor. Power is generated at 2,400 v., 3 phase, 60 cycles and delivered to the main switchboard, where it is distributed by circuits. Transformer stations varying in size from 600 to 1,000 kva. are located around the plant to reduce the voltage to 440 v. for most operations. An average power factor of 87 percent is maintained by use of synchronous motors and capacitors.

Electrical and mechanical repair shops are provided both underground and on surface. The underground shop handles all ordinary repairs on mining equipment, only such jobs that require machine work being taken on top.

Due to the isolation of the plant, it was necessary to provide on surface a large machine shop equipped with facilities capable of handling all major mechanical and electrical work promptly. Consequently, almost any repair job can be done with minimum delay to operations. Provision was also made to fabricate most of the plate and structural steel used in structures and equipment.

Pipe cutting and threading is handled by machinery and electric and gas welding is used extensively.

Supplies are purchased in carload lots where feasible and large stocks are maintained at all times. It is necessary also to carry in stock ample quantities of both electrical and mechanical repair parts, to assure against costly and annoying delays.

Mammoth-St. Anthony's Complex Operations

Five products are recovered from a low-grade gold ore

Panorama of the Mammoth-St. Anthony plant at Tiger, Ariz.

John B. Huttl
Assistant Editor

AT TIGER, Pinal County, Ariz., Mammoth-St. Anthony, Ltd., is conducting a mining operation of unusual interest. Here, five products—gold, silver, lead, molybdenum, and vanadium—are being recovered from a relatively low-grade gold ore, and one of the smallest smelters known is in operation. The enterprise is essentially a low-grade gold mine, with a large daily tonnage and careful management as the principal requisites for making the orebodies economically important. Since the European market for molybdenum and vanadium was lost at the outbreak of the war more than 12 months' output of these commodities has accumulated at the mine, resulting in a tie-up of capital from this part of the operation.

The camp, formerly known as Schultz, now has a population of nearly 1,700 and is in the Mammoth or Old Hat district, about 50 miles northwest of Tucson and 25 miles south of Winkelman, the nearest rail point. The village of Mammoth, on the San Pedro River, where early operators maintained stamp mills, is but 3 miles distant. Gravel roads run to Tiger and to the mines. Elevation is about 3,120 ft., and the climate is subtropical and dry.

Mining dates back to 1879, when Frank Schultz located the Hackney and Aaven claims on the Collins vein. Location of the Mohawk claim and the Mammoth and Mars claims followed in 1881 and 1882. Since then the mines have been worked intermittently by various owners, and are credited up to the end of 1936 with a production exceeding $5,000,000. The increase in the price of gold to $35 in 1933 resulted in a revival of mining. During that year Molybdenum Gold Mining Co., a subsidiary of Molybdenum Corporation of America, acquired the Mohawk and New Year mines, and began development work, including completion of a concentration-cyanide mill projected by a previous owner. The present operators entered the district in 1934. Mammoth and Collins mines were reopened, and after rehabilitation and development directed by the late Foster S. Naething, production began in 1935, the ore being sent to the mill of Molybdenum Gold for treatment. Mammoth-St. Anthony purchased this mill in 1937. Its capacity has been gradually increased from 90 to 560 tons by adding extra equipment. Current holdings, both owned and leased, include 19 patented claims and 19 unpatented claims.

Local geology is complex. The country rock is composed of irregular intrusions of granite and rhyolite, through which the veins pass without change in values. Veins are brecciated shear zones, having no definite walls and no vein quartz, but some silicification of rhyolite and granite. They strike west-northwest and dip steeply southwest. Gangue is brecciated country rock, cemented and replaced by quartz and calcite, with some barite and fluorite. In general, the gold is fine and seldom visible. It is the most valuable constituent, the major portion occurring in dense greenish-yellow quartz. Minerals found in the oxidized zone include limonite, hematite, anglesite, cerussite, linarite, bronchantite, malachite, azurite, calamine, wulfenite, vanadinite, descloizite, cuprodescloizite, pyrolusite, and others. Galena, sphalerite, and some molybdenite occur as hypogene minerals. Ore depositions appear to be along sections of the faults where shattering and brecciation were intense. In consequence, vein or lode structures are very irregular, stoping widths varying from a few feet up to 60 ft. in one place. Important widening of the veins is caused by strike faults at slightly steeper dip than the vein itself, which in most stopes have doubled or tripled the original width of vein. Mineralization is spotty, requiring much development and sampling. Wall-rock alteration along the veins consists mainly of chloritization and silicification. The principal veins currently exploited are the Collins and Mammoth. Each is about 1,000 ft. long and averages 18 ft. in width. A general layout of the vein systems and the main fault, known as the Mammoth, is given in Fig. 1. In

the Collins vein, oxidation is complete at about 600 ft., and in the Mammoth at 800. The latter is cut off by the Mammoth fault between 850 and 900 ft. As the sulphides carry practically no gold, profitable operation ceases at these depths.

Current production comes almost entirely from the Mammoth property, including the Collins and Mammoth vein systems. This is opened through the 835-ft., three-compartment Mammoth shaft equipped with a steel headframe, circular steel bins for ore and waste, and 1½-ton skips actuated by a 150-hp. double-drum hoist. All levels below the 700 are served by an incline winze. Thus, ore mined on the 760 and 800 levels is hoisted through the winze and then hand-trammed to the Mammoth shaft. A 1-ton skip is used, operated by an air hoist. The level interval is 100 ft. The 760 level was the deepest, or water, level prior to purchase by the present operators. At Mammoth and Collins mines the workings are connected by crosscuts on the 300 and 500, and by a crosscut on the 700. No. 2 shaft, near the Mammoth shaft, facilitates supply of material to underground points, and ventilation. Development crosscuts and drifts are advanced 5x7 ft. in cross-section, with the ordinary drill round consisting of 15 4-ft. holes. Drilling is done with modern air machines using ⅞-in. hollow-hex. steel with forged-on bits. Trojan 40 percent ammonia powder is used in conventional blasting. In shaft work only gelatin dynamite and electric primers are employed. Fuse firing is practised in stopes, raises, and drifts. As the ground around the larger stopes usually is heavy, removal of blocked-out ore is done by both square-set and cut-and-fill rill methods, with most square-set laid out in rill fashion. The 8x8-in. timbers used are shipped from Arizona lumber camps. Material for stope filling is obtained from surface glory-holes. In the area worked by former operators by open stopes and which caved in 1892 to the 400 level, and in 1902 to the 760 level, recovery of pillars is under way. The section is 350 ft. long.

Rock from development is loaded into 1-ton end-dump cars by Eimco loaders. Mancha locomotives are used on the two haulage levels mentioned. Elsewhere hand-tramming is practiced. Scrapers are used intermittently in some development work. Water is handled by a 600-g.p.m. Pomona deep-well pump discharging into a sump on the 700 level, which also receives that from a pump on the 800 level, whence it is pumped to surface by two 500-g.p.m. I.-R. centrifugal pumps. A second pump installation consists of a centrifugal pump which delivers water from the incline winze to a triplex pump on the 700 level, this in turn discharging into a surface tank. Ventilation is natural. Ore hoisted in skips through the Mammoth shaft is dumped into a circular steel bin which is the feed point for the aërial tram that leads to the crushing plant near the Mohawk shaft, more than 1,200 ft. away. The buckets hold 400 lb. each.

At the Mohawk and New Year mines, which explore some portions of the Mammoth vein, activities are confined to development. The three-compartment Mohawk shaft, served by a 100-hp. double-drum hoist with balanced cages, is being deepened, and was 50 ft. below the 650 level at the time of my visit. Only 140 ft. distant is the New Year shaft. It has two compartments, and is not in use at present. Water in the Mohawk shaft and drainage from the 650 level, corresponding in elevation to the 800 level in the Mammoth mine, is handled by an I.-R. station pump. Ventilation as at the Mammoth mine is natural. Total water made by all mines ranges from 600 to 700 g.p.m. With oxidation complete to the 800 level in the Mammoth mine and to the 500 level in the Mohawk mine, all ore sent to the mill is oxidized. Mine samples are taken for gold only, which is the main value.

Ore received at the mill is of low grade, hard, and besides the gold contains commercial quantities of lead, silver, molybdenum, and vanadium. Gold is recovered as precipitate from cyaniding and partly in concentrates produced by the tabling and flotation. These products are trucked to the near-by smelter, where they are fused with sodium carbonate and lixiviated. Final shipping products are lead bullion containing gold and silver, and crude salts consisting chiefly of sodium molybdate and vanadate. Ore treatment is shown in the flowsheet in Fig. 2. As will be seen, two-stage crushing, grinding to table size, tabling, regrinding of table tailings, flotation of reground table tailings, and cyaniding of flotation tailings are involved. Present production of ore is about 560 tons a day. The 6-in. mine-run ore is brought to the 500-ton storage bin at the crushing plant via a 1,200 ft. aërial tram equipped with 400-lb. buckets and is crushed to 1½ in. in a No. 4 McCully gyratory, followed by a 2x4-ft. Robins screen in open circuit and fitted with steel plates containing ½-in. round holes. Screen oversize is re-

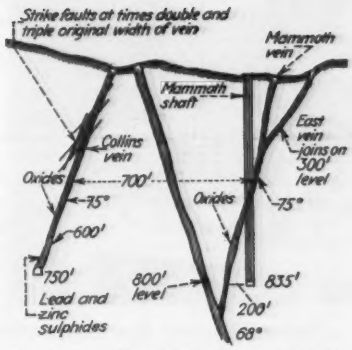
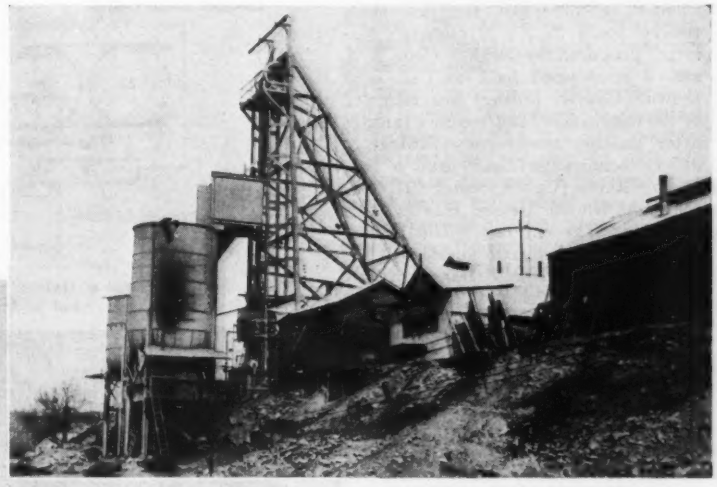
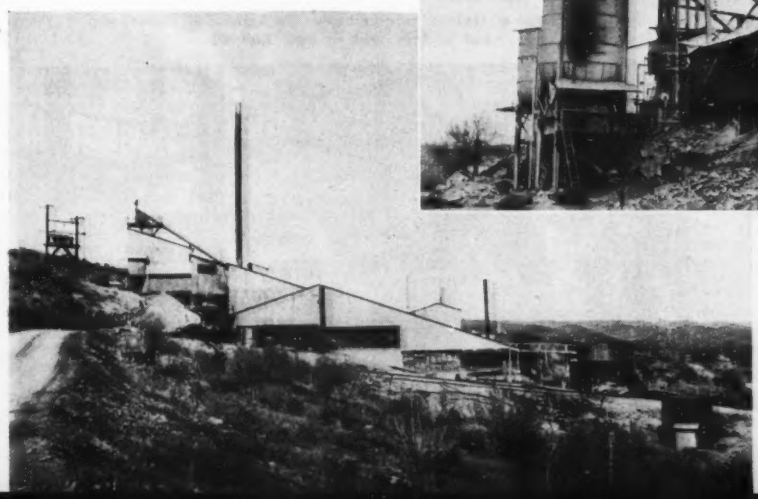


Fig. 1 . . . General layout of the vein systems and the main fault, known as the Mammoth, in vertical projection



Surface plant at the Mammoth shaft. From the steel bin the aerial tramway buckets are loaded



A general view of the 20-ton smelter

Pouring 90-lb. lead pigs at the smelter

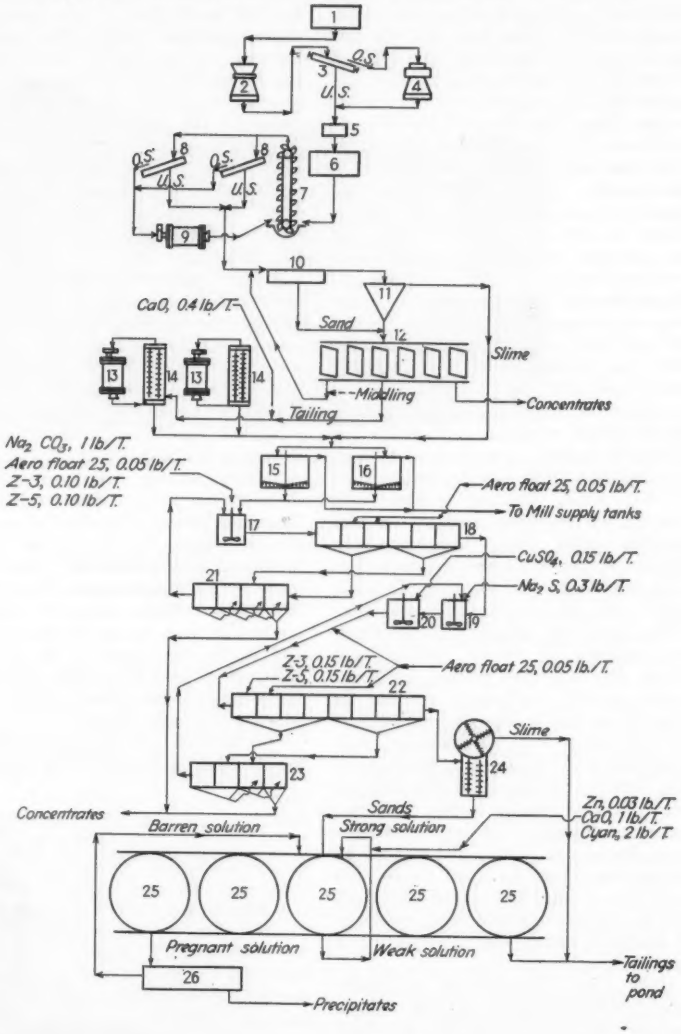
duced to ¼ in. in a 3-ft. Symons cone crusher. This product and screen undersize go to a 500-ton bin.

From here the ore is elevated to two Leahy screens, the oversize going to a 5x10-ft. Marcy mill in closed circuit, and the 6-mesh undersize flowing to a St. Joe classifier. The overflow from this unit passes to a cone classifier, and the sands as well as those from the cone are treated on six Wilfley tables. Slimes from the cone flow to two thickeners at the head of flotation. Percentages of values recovered in the table concentrates are as follows: Lead, 35; molybdenum, 50; vanadium, 35; silver, 10; and gold, 15. Table middlings are returned to the St. Joe classifier, and the tailings are directed to the regrind section. Here, two No. 64½ Marcy ball mills and two 6-ft. Dorr classifiers are available. The 48-mesh overflow is thickened in two Dorr thickeners before entering the 5-ft. gold conditioner at the head of the six No. 21 Denver Sub-A gold roughers. Thickener overflow is pumped to the mill water supply tanks. Cleaning of the products of the gold roughers is done in four No. 18 Denver Sub-A cells, with the overflow of the first three rougher cells going to No. 1 cleaner and that of the last three rougher cells to No. 2 cleaner. Material produced by cleaner Cells 2, 3, and 4 is stepped back to Cell 1 as shown. Cleaner tailings are returned to the conditioner, and rougher tailings after passing successively through a 3x5-ft. conditioner(Na_2S) and a 7x7-ft. conditioner($CuSO_4$) enter Cell 1 of the eight No. 21 Denver Sub-A oxide roughers. The product made by the first four rougher cells goes to Cell 2 of the four-cell No. 18 Denver Sub-A cleaner unit, and that of the last four rougher cells enters Cell 3 of the cleaner. The combined overflow from cleaner Cells 3 and 4 is stepped back to Cell 2, and that from Cell 2 to Cell 1, which produces the final concentrate containing the following percentages of recoverable values: Lead, 35; molybdenum, 40; vanadium, 35; silver, 10; and gold, 65. Rougher tailings go to a large bowl classifier making a sand-slime separation for the cyanide sand-leaching plant, and cleaner tailings are returned to the 3x5-ft. conditioner.

LEGEND

1. 500-ton coarse-ore bin
2. No. 4 McCully crusher
3. 2x4-ft. Robins screen
4. 3-ft. Symons cone crusher
5. Merrick weightometer
6. 500-ton fine-ore bin
7. Bucket elevator
8. Leahy screens
9. 5x10-ft. Marcy ball mill
10. St. Joe classifier
11. Cone classifier
12. Wilfley tables
13. 64½-in. Marcy ball mills
14. 6-ft. Dorr classifiers
15. 60-ft. Dorr thickener
16. 40-ft. Dorr thickener
17. 5-ft. gold conditioner
18. No. 21 Denver Sub-A gold roughers
19. 3x5-ft. Na_2S conditioner
20. 7x7-ft. $CuSO_4$ conditioner
21. No. 18 Denver Sub-A cleaners
22. No. 21 Denver Sub-A oxide roughers
23. No. 18 Denver Sub-A cleaners
24. Dorr bowl classifier
25. 500-ton sand-leaching tanks
26. Precipitation plant (Merrill-Crowe)

Fig. 2 . . . Flowsheet of ore treatment in the Mammoth-St. Anthony mill



Firing end of 11x16-ft. reverberatory. The furnace is center-fed and is fired with a light fuel oil

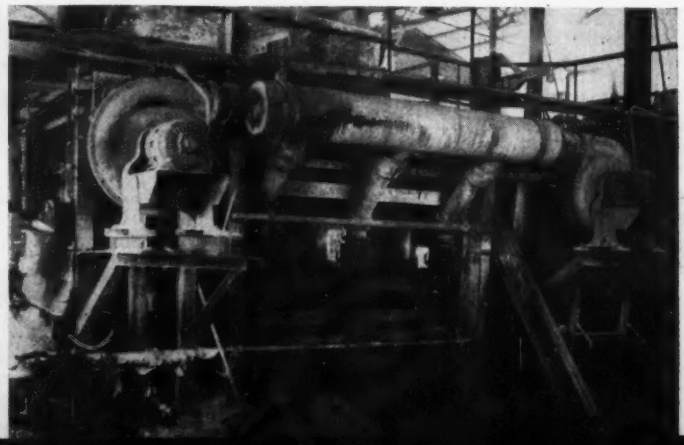
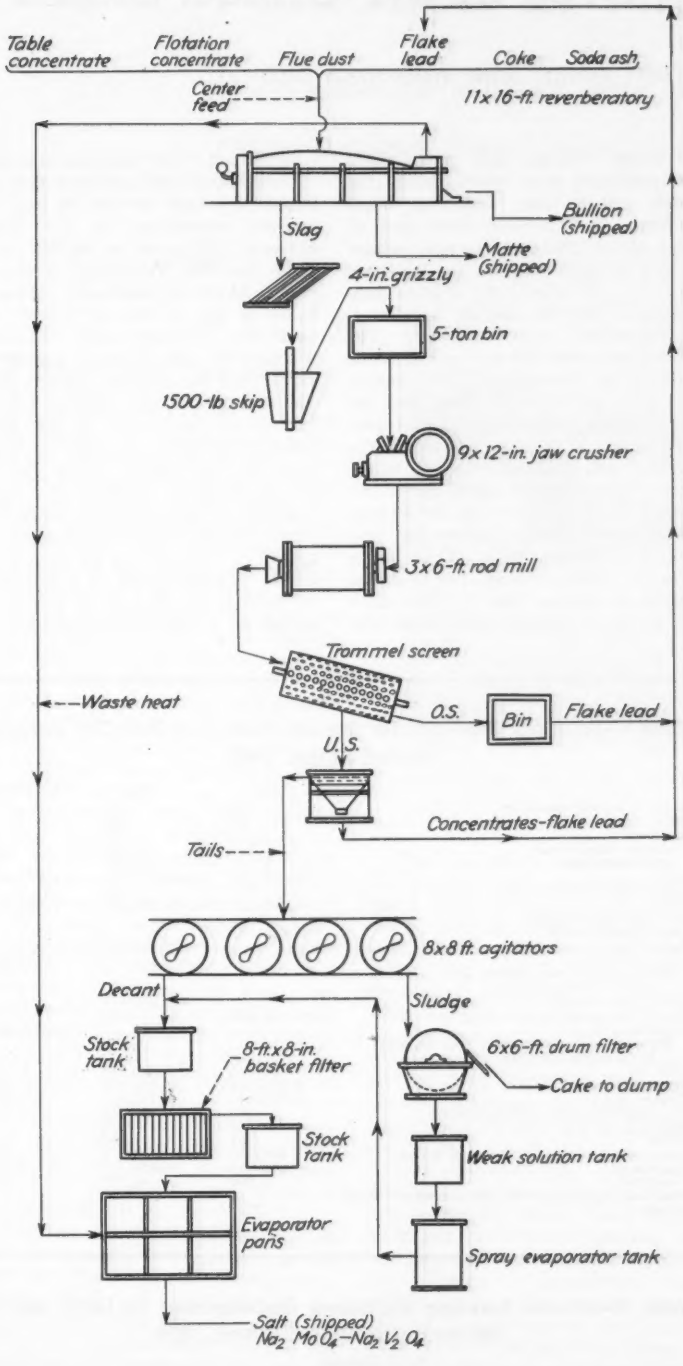


Fig. 3 . . . Flowsheet of the smelter and the slag-treatment plant



The kinds and amounts of reagents used per ton and points of addition are given in the flowsheet. Ratio of concentration is 40:1, power consumption 18 kw.-hr. per ton, and water consumption 500 gal. per ton. Table concentrates and flotation concentrates are trucked to the nearby smelter for further treatment.

The sand-leaching plant is of conventional design. To treat the plus-200-mesh sands separated by the bowl classifier, which constitute about 65 percent of the total mill tonnage, there are available five 500-ton leaching tanks. Operation is on a six-day cycle—that is, filling, leaching, and removal of leached sands. Pregnant solution is treated by Merrill-Crowe precipitation. Precipitates, representing 12 and 15 percent of the gold and silver in the ore, are shipped in sealed cans to the American Metal Co., Carteret, N. J. Leached sands and bowl classifier slimes are pumped to the tailings pond.

Capacity of the smelter is about 20 tons per day. Operations are explained in Fig. 3. Smelting is done in a 11x16-ft. reverberatory designed for center feeding, and equipped with three low-pressure burners using light fuel oil. Fuel consumption is about 100 gal. per ton. A typical charge is made up of 12 tons table concentrates, 12 tons flotation concentrates, 300 lb. flue dust, 300 lb. flake lead, 1,440 lb. coke, and 4,400 lb. soda ash. Charges are fed to the furnace at 30-min. intervals. Matte and lead in the form of 90-lb. pigs are shipped to U. S. Metals Refining Co., Carteret, N. J., and the slag containing the molybdenum and vanadium values goes to the treatment section producing Na_2MoO_4–$Na_2V_2O_4$ salt, which is shipped in bulk in box cars to the Molybdenum Corporation of America at Washington, Pa. At the slag-treatment plant the material is passed over a 4-in. steel grizzly, elevated in a 1,500-lb. skip to a 5-ton bin, and crushed in a 9x12-in. jaw crusher. The product is ground in a 3x6-ft. rod mill, the discharge passing to a trommel. Oversize or flake lead goes to storage, whence it is withdrawn periodically to make up furnace charges, and the undersize receives further treatment in a 12x16-in. Harz jig. Concentrates are returned to the flake-lead pile on the charge make-up floor, and tailings go to four 8x8-ft. agitators. The sludge flows to a 6x6-ft. drum filter, with the cake going to the residue dump and the filtrate passing successively to a weak-solution tank, spray-evaporator tank, and stock tank receiving the decant from the agitators. From the stock tank the solution flows to a 8-ft. by 8-in. basket filter, and then to a stock tank feeding the evaporator pans producing the Na_2MoO_4–$Na_2V_2O_4$ salt. These are heated by the reverberatory waste-heat gases.

Over-all recoveries are as follows: Gold in mill, 86 percent; in smelter, 93 percent; lead in mill and smelter, 57 percent; molybdenum in mill, 97 percent; in smelter, 74 percent; and vanadium, in mill and smelter, 27 percent. Power is obtained from Coolidge dam. Transformers at Tiger step the current down to the desired voltages. Other structures include repair shops, change house, general office, warehouse, laboratory, guest house, hospital, and staff dwellings. J. L. Fozard is general manager, to whom thanks are due for the many courtesies shown me durring my recent visit to Tiger. Members of his staff include Warren Howes, mill and smelter superintendent; J. A. Richards, mine superintendent; E. B. Daggett, geologist; Clyde Bilson, master mechanic; and H. J. Nickolds, smelter foreman. Henry C. Carlisle, San Francisco, is consulting engineer.

Lead and Zinc Ores in the United States

Census report reveals some interesting facts

BASIC CONDITIONS in the lead- and zinc-ore industries of the United States in 1939 have been outlined in a publication issued by the Bureau of the Census, revealing a number of interesting trends and characteristics. Similar data on copper were reviewed in *E.&M. J.*, May, 1941.

The figures given in the accompanying text and tables relate to mines and mills producing ores and concentrates valued chiefly for their lead or zinc content. It is estimated that these mines and mills accounted for 87 and 91 percent, respectively, of recoverable lead and zinc contained in all ores and concentrates produced in the United States in the year reviewed. The remainders were produced by operations engaged in producing ores and concentrates valued chiefly for silver, gold, copper, fluorspar, manganese, and tungsten.

The number of producing zinc and lead mines declined from 473 in 1919 to 375 in 1929, and 254 in 1939. This reduction resulted largely from consolidation of single properties into larger operating units, exhaustion of ore at some properties, and temporary suspension due to unfavorable economic conditions at others. The 254 mines and 118 ore-dressing mills were operated by 191 companies and provided employment for 15,544 wage earners and 1,470 salaried workers.

Average recoverable metal content per ton ore produced at lead mines, and sold or sent to smelters or ore-dressing mills, for the United States as a whole, was 83 lb. of lead and 18 lb. of zinc, in addition to substantial quantities of gold, silver, and copper. In southeast Missouri recoverable metal (all lead) averaged 62 lb.; in Idaho, 144 lb. of lead and 60 lb. of zinc. Average recoverable metal content per ton of ore produced at zinc mines and sold or sent to mills or smelters was 56 lb. zinc, 7 lb. lead, and some gold, silver, and copper. In Oklahoma, leading zinc-mining State, recoverable metal per ton of zinc ore was 32 lb. zinc and 4 lb. lead.

Almost all of the ore mined at lead and zinc mines in the United States in 1939 required concentration before smelting. Of the total of 16,263,000 tons of ore mined (excluding tailings), only 190,000 was direct-smelting ore—chiefly from mines in Utah and Idaho. Crude ore and tailings concentrated during 1939 at lead and zinc mills (including custom mills that treated some ore from mines that were too small to come within the scope of the census canvass and from mines that produced ores valued chiefly for metals other than lead or zinc) amounted to 23,998,000 short tons, of which about 7,000,000 tons was tailings treated at zinc mills in the Tri-State district. Of the ore and tailings treated, 1,960,000 tons, or 8 percent, was recovered as concentrates. The ratio of concentration for lead and zinc mills in the Mississippi Valley region was considerably lower than that for other regions, averaging about 4 percent, compared with 22 percent in the Western and Eastern regions, where the ore mined was of higher grade. In each of the regions, the ratio of concentration was slightly higher for lead mills than for zinc mills. The average ratio of concentration for material treated at custom lead and zinc mills was almost 13 percent, compared with 7 percent at other lead and zinc mills.

Value of all products per man-hour worked by wage earners at lead- and zinc-ore operations in the United States in 1939 averaged $1.94. It was $1.96 for the Mississippi Valley region; $2.73 in southeast Missouri; $1.63 in the Tri-State district; and $1.52 in Kentucky and Wisconsin. Average for the Western region was $1.95; for the Eastern region, $1.84. The average of $2.24 per man-hour for lead mines and mills in the United States was about 31 percent higher than the average of $1.71 for zinc mines and mills. Mining of a ton of crude ore at lead and zinc mines in the United States in 1939 required an average of 1.5 man-hours of wage-earner labor. Concentration of a ton of ore or tailings at lead or zinc mills required an additional 0.3 man-hour.

Table I—Principal Statistics for the Lead-Ore and Zinc-Ore Industries, United States, 1939

	Lead	Zinc	Lead and Zinc
Number operating companies	58	140	191
Number mines	68	186	254
Number mills	26	92	118
Av. number wage earners	6,889	8,655	15,544
Av. hourly earnings	$0.71	$0.56	$0.62
Wages	$9,820,575	$10,226,679	$20,047,254
Number salaried employees	676	794	1,470
Salaries	$1,765,352	$1,881,683	$3,647,035
Recoverable metal content			
Lead in lead ores, lb.	598,526,781	.....	.....
Lead in zinc ores, lb.	.....	122,316,870	.....
Total lead, lb.	.....	.....	720,843,651
Percent total recoverable lead reported by U. S. Bureau of Mines	.....	.....	87.1
Zinc in zinc ores, lb.	.....	934,266,616	.....
Zinc in lead ores, lb.	131,562,486	.....	.....
Total zinc, lb.	.....	.....	1,065,829,102
Percent total recoverable zinc reported by U. S. Bureau of Mines	.....	.....	91.3
Gold, oz.	39,050	20,888	59,938
Silver, fine oz.	9,209,732	2,073,086	11,282,818
Copper, lb.	10,277,418	1,133,081	11,410,499
Av. number equivalent full days active	260	230	242
Mines	260	224	240
Mills	262	246	250
Electric energy consumed, thousand kw.-hr.	250,250	248,512	498,762
Purchased	245,645	204,062	449,707
Generated	4,605	44,450	49,055
Horsepower rating of power equipment per wage earner	27.8	17.4	22.0

Table II—Power Loading Machines Underground in Lead and Zinc Mines in the United States, 1939

	Shovel Loaders requiring a minimum working height of 8 feet or less			Requiring a minimum working height of more than 8 feet	Scraper Loaders and Slushers		
		Kind of Power used				Kind of Power Used	
Region and Kind of Mine	Total	Electric	Compressed air		Total	Ekectric	Compressed air
United States, total	132	85	47	4	285	202	83
Lead mines	111	84	27	1	90	42	48
Zinc mines	21	1	20	3	195	160	35
Mississippi Valley region, total	84	84	..	..	75	70	5
Lead mines	84	84	..	..	38	38	..
Zinc mines	...	..	..	..	37	32	5
Western region, total	38	1	37	4	92	39	53
Lead mines	27	..	27	1	52	4	48
Zinc mines	11	1	10	3	40	35	5
Eastern region	10	..	10	..	118	93	25

Placer Valuation in Alaska

By the application of certain factors, adapted from prospecting in frozen ground to adjacent thawed ground, under-runs and over-runs are controlled

L. C. Doheny
Fairbanks, Alaska

United States Geological Survey

Dog teams serve the Alaskan miner and prospector in undertaking his winter work

THROUGHOUT the interior of Alaska most gold placer deposits are in places where the ground is permanently frozen to a greater depth than bedrock. The ground permanently frozen is prospected by open churn-drill holes; the volume sampled in the pay horizon determines the value of the ground to be mined. In the earliest prospecting of frozen ground an effort was made to adapt the Radford and Keystone factors, which are on the area basis, to open holes in frozen ground. Size of bit was taken as the determining factor for the hole area. This early proved faulty. With later development, hole measurement by water displacement demonstrated that bit size has no relation to hole area.

Prospecting in frozen virgin ground has been reliable to an astonishing degree, but in thawed ground the recovery over estimate is seldom 100 percent. Though an over-run is always welcome, it shows that the economic limits of the mining ground have every probability of being incorrect; therefore, workable ground has not been prepared for mining. The minable ground, once passed by, cannot be worked separately, but must be prepared and mined with the whole body or not at all.

By applying certain factors, adapted from the prospecting in frozen ground to the adjacent thawed ground, definite control on under-runs and over-runs is established.

Hole in Frozen Ground: Volume Measurement—Reference to the sample drill log (Fig. 1) for a hole in frozen ground, which gives the data as recorded, shows that the column headed "Measured Volume Loose" records the cubic-foot measurement to the nearest 1/1,000. This is a measurement of the drill cuttings that have been washed completely free of all clay and are ready for panning. These are washed to the same standard that will be used in holes in thawed ground.

A bucket with a diameter of about 6 in. of suitable accuracy for measuring volume is chosen. This size will give a good vertical rise for measuring where the volume is small.

Water is used to calibrate this volume bucket. With the fact as a basis that 1 cu.ft. of water weighs 1,000 oz. avoirdupois, one ounce will be 1/1,000 cu.ft. Taking 0.005 or 0.010 cu. ft. at a time, we note the displacement in the bucket and mark it inversely on the measuring stick.

To measure the volume of the cleaned cuttings in the bucket, the average of the depths on two opposite sides of the bucket is taken. This is recorded on the drill log under "Measured Volume Loose."

Pay Horizon—The pay horizon through which the water measurement is to be taken is selected as that below 32 ft. and down to and including 41 ft. (as recorded under "Depth of Pumping or Casing" in the aforesaid sample drill log). Part of the gold from one drilling and pumping operation always tends to be recovered upon subsequent pumping. Common practice is to include the upper horizon showing only a few colors and to disregard the lower horizon showing the same amount.

Water Measurement—Reference is now made again to the drill log in Fig. 1. In the columns to the right, and reading up the log, are the records of the water measurement. Start is made at the bottom of the hole, the first operation being to determine the amount of seepage. This is the small amount of water coming into the hole from near the surface, at the junction of permanent frost and seasonal thaw.

It will be noted that during the initial interval the seepage was 3 in. The next step is to pour in 0.25 cu.ft. of water and note the rise, which is 15 in. The seepage in the initial 10-minute interval is then taken as 3/15 $\times$ 0.25 cu.ft., or 0.050 cu.ft.

The water measurement is completed, adding 0.25 cu.ft. of water during each interval, the rise being noted each time. The lower end of the hose into which the water is poured must be kept below the water level of the column being filled, to avoid sloughing of gravel in the upper parts of the hole. This would give a faulty volume for measurement. The most practical way to prevent such sloughing is to use ice and ice water during the drilling to keep the temperature of the hole at 32 deg. F. For the same reason, ice water is used in the water measurement.

Inasmuch as the pay horizon has its top at 32 ft., as stated, the water measurement is continued at intervals, as recorded, until finally one of the regular additions of 0.25 cu.ft. of water brings the total height of the water column to something above 32 ft. Then the seepage at the top of

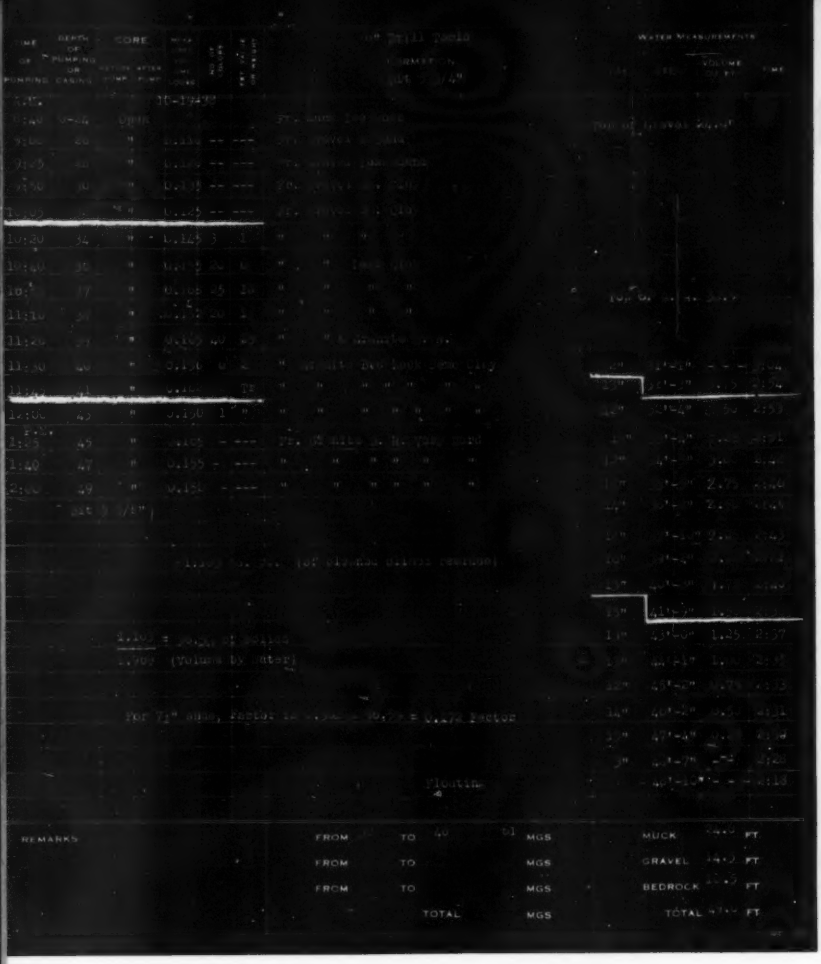


Fig. 1 . . . Part of a drilling log kept for a hole in frozen ground. The items shown are reproduced from a printed form, the upper and lower parts of which have been omitted

the hole is determined for a second 10-minute interval and is found to be 2 in. Interpolating this against the column size adjoining, there are 2/13 $\times 0.25 = 0.035$ cu.ft. for the seepage at the top. The average of the two seepages, top and bottom, is 0.042 cu.ft. per 10 minutes.

The next step is to determine the volume in the pay horizon. The *cumulative* volume at 41 ft. 9 in. was 1.50 cu.ft. This is 9 in. below the bottom of the pay horizon, so 9 in. of this part of the column is *subtracted* or $9/13 \times 0.25$ cu.ft., or 0.173 cu.ft.

At the top, at 31 ft. 3 in., the correction back to 32 ft. is 9 in. of a 12-in. rise, or $9/12 \times 0.25 = 0.187$ cu.ft. This is to be subtracted. Now the total horizon between 31 ft. 3 in. to 41 ft. 9 in. has a volume of $3.75 - 1.50$, or 2.25 cu.ft. From this last volume are taken the interpolations at top and bottom—a total of 0.360 cu.ft. The total then is 1.890 cu.ft. of water poured into the pay horizon, but the seepage from the top is added to the water measurement during the time from 2:38 to 2:53, or 15 minutes.

For a 10-minute period the seepage averaged 0.042 cu.ft. Multiplying 0.042 by 1.5 (a 15-minute period) gives a result of 0.063 cu.ft., which is added to the total volume in the horizon, or 1.953.

Calculations. The value is then:

$$\text{Value/cu.yd.} = \frac{\text{Milligrams x value in cents per mg.}}{\frac{\text{cu.ft}}{27}}$$

$$= \frac{\text{Mg. x cents per mg. x 27}}{\text{Cu.ft.}}$$

Substituting we get

$$\frac{61 \text{ mg. x } 0.0966\text{¢ x } 27}{1.953} = 81.4\text{c/cu.yd.}$$

(fineness of gold 860 = 0.0966c/mg.)

$$81.4 \text{ x } \frac{9 \text{ (pay horizon)}}{17 \text{ (mining section)}} = 43.1\text{c/cu.yd.,}$$

mining section

$$43.1 \text{ x } \frac{17}{27} = 27.1\text{c/sq.ft. on bedrock}$$

The total solids recovered from the pay horizon, or the clean, slimed residue, was 1.103 cu.ft; and the total volume sampled in the same horizon was 1.969 cu.ft. Therefore, $\frac{1.103}{1.953} =$ 56.5 percent.

The "thawed" hole adjacent to this hole was drilled with 6-in. casing which uses a 7½-in. shoe. The area of a 7½-in. shoe is taken at 0.306 sq.ft. Therefore, a 1-ft. drive should recover the solids in proportion to 0.306×56.5 percent $= 0.172$ cu.ft. per average linear foot of drilling in the pay horizon.

Column Corrections — The water measurement shows there are "bellies" and throats in a hole in frozen ground. Often the loosest layers that show the larger volumes contain more gold. It is never exactly known that the bellies attain their full size at the time that particular horizon is being drilled. Often the boxes on the drill stem cause sloughing 5 to 6 ft. above the actual prospecting depth. In this case the gold and cuttings are recovered at an horizon lower than the one in which they occur.

Placer engineers are not in agreement on any one method of calculating corrections for bellies and throats in the pay horizon. It has been found inaccurate to cut down the former to average and to make a deduction in the gold recovered.

Hole in Thawed Ground—Over-runs and Under-runs—In drilling thawed ground there are two opposite cases. Both can give the value incorrectly. The first extreme case is where there is loose, thawed ground with an excessive void space. The best example of this would be loose, rocky, dredge tailings entirely lacking the finer sizes. A regular drive of the casing shows a normal core. But when the rocks are drilled to the size that will allow the bit to register at the depth of the shoe face, the void space in the tailings has found room for the drill cuttings from above. The churn-drill action tends to pound the gravel down through the shoe. The result is that, in the loose ground, the pump is able to get little or no recovery.

The other extreme is encountered when a drill hole breaks through frost into the thawed pay horizon, or through a tight layer into a zone of artesian pressure. In this case an excess core is pushed up into the casing. Formerly the excess core was corrected to a normal one. This was found unsatisfactory, as there was no check on how much additional gravel came into the casing while drilling and pumping the excess core. The hydrostatic pressure below might or might not have added more gravel before that particular horizon was passed. The over-run is found with a surprisingly high accuracy by using the volumetric check on the cleaned solid residue.

Calculation—Referring to the sample log for drilling in thawed ground the pay horizon is taken at 10 to 17 ft. The loose volumes of cleaned solids in this horizon total 1.226 cu.ft. Then deducting the night cap, which was the protection against salting, the remainder is 1.126 cu.ft.

From the hole in frozen ground the cleaned solids should have been 0.172 cu.ft. for the average foot through the pay horizon, or $7 \times 0.172 = 1.204$ cu.ft.

This ($\frac{1.126}{1.204}$ = 93.4 percent) shows a slight under-run. The 74 mg. of gold are corrected to 100 percent or 79 mg. The calculations are then similar to those for a hole in frozen ground.

In the case of an over-run of cleaned solids, the excess would be found and the recovered milligrams would be cut down proportionately.

To make a correction for an over-run or under-run in any particular horizon is an art. The most experienced engineers make these interpretations, but they consider the possibility that the gold might be recovered at a lower horizon than its natural occurrence in the thawed gravel. There is always the possibility that an artesian pressure will blow more of the lighter gravel into the casing than of the heavier black sand and gold, which remain at the bottom of the hole or below it.

It is highly probable that the deeper drilling section below the pressure horizon contains gold from the upper loose gravel that settles ahead of the drill shoe, and is recovered when a tighter layer such as the top of bedrock is reached. By using lead shot, short pieces of copper wire, and similar material, it was found that only a partial recovery can be expected through the loose gravels; but frequently up to 80 percent of the shot has been recovered when the shoe reached bedrock or a tighter layer.

Also, it must be remembered that the percent of solids can change from the center of a pay streak to its sides. There is also the probability of a change longitudinally in the same pay streak. Thus the factor should be taken from the nearest holes in frozen ground and applied to the nearest thawed areas.

Application — Calculations of the over-runs and under-runs from placer drilling, based on the cleaned, slimed solids factor, have given recoveries that check with the estimates to a much happier degree than by former methods. The valuation from the cleaned solids factor has been between 80 and 120 percent compared to the Radford and Keystone factors, which have seldom shown the recovery-estimate at 100 percent and have been in disagreement 25 to 500 percent.

The first question to arise is: How can this different method be applied where there is no frozen ground from which a constant can be derived? The answer is: If the project warrants the expenditure, the ground could be frozen artificially and kept frozen while an open hole is drilled. This method would be much cheaper than sinking shafts and would also eliminate the errors that come with wet shaft sinking.

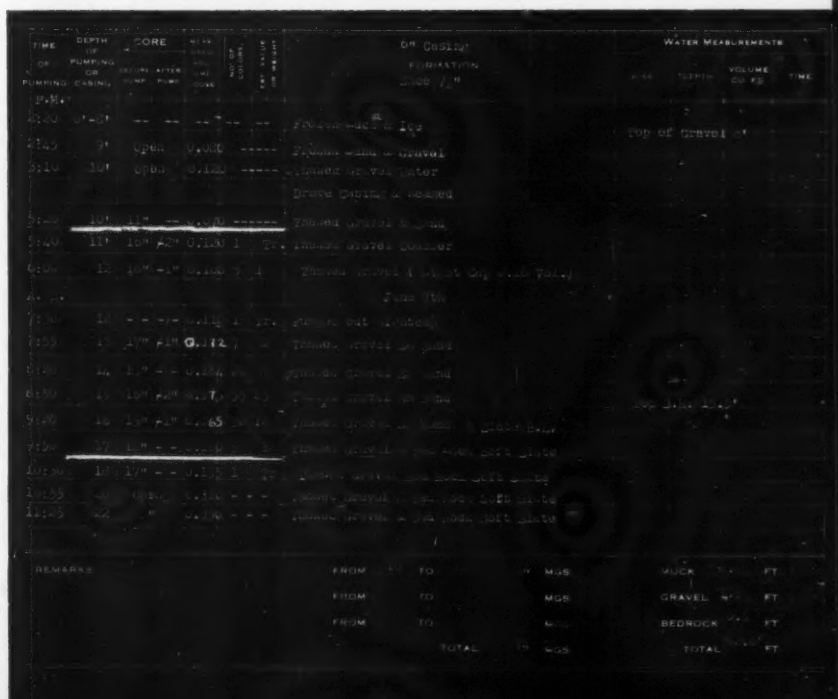


Fig. 2 . . . Part of a field log kept for a hole drilled in thawed ground. As in Fig. 1, the upper and lower parts have been omitted

Mica-Bearing Pegmatites

In Spruce Pine District, North Carolina

THE MICA DEPOSITS of the Spruce Pine district, western North Carolina, have been examined by T. L. Kesler and J. C. Olson, geologists, as a part of the investigation of domestic deposits of strategic minerals by the Geological Survey, U. S. Department of the Interior. The examination was prompted by the current demand for sheet mica of highest quality for radio equipment, airplane spark plugs, transformers, and generators. Most of the mica used for these purposes heretofore has been imported, principally from India.

In the Spruce Pine district, mica and feldspar have been mined from more than 500 pegmatite bodies, which are distributed over an area of 250 sq.mi. The pegmatite bodies, some of which are more than a hundred feet thick, are most closely associated with a variety of granite called alaskite and occur both in the alaskite and in adjacent metamorphic rocks. Mica in commercial quantities commonly is localized in shoots, called "streaks," within the pegmatite bodies. The shoots are irregular in shape, and the mica crystals, called "blocks," are distributed unevenly in them. No adequate method of prospecting in advance of mining has been developed.

At least 16,600,000 lb. of sheet and punch mica have been shipped from this district since 1868, when mica mining began. It is estimated that the part of this output classed as domestic clear-sheet mica averaged about 45,000 lb. per year during the period 1924-40. Analysis of the recorded past and present output of 131 mines indicates that a considerable proportion of the mica probably was equivalent for commercial purposes to the higher grades of imported mica, although it is impossible to trace all uses to which mica from the Spruce Pine district has been put. It is concluded that a more systematic planning of mica mining in the district as a whole could easily raise the output of relatively clear sheet mica to 90,000 lb. per year, and perhaps to several times this amount.

Among the properties affecting the usefulness of mica is the so-called power factor, or the loss of electrical energy in films used as the dielectric in condensers. Through a cooperative arrangement, E. L. Hall, of the National Bureau of Standards, made careful measurements of the power factors of 196 samples of sheet mica obtained by the Geological Survey from 109 mica mines and 15 feldspar mines. Samples from 71 mines in the group were found to have power factors within the limits allowed for mica used in radio-transmitter condensers. Samples from ten mica mines in North Carolina, six of them from mines in the Spruce Pine district, were submitted to two commercial users for tests to determine their utility in condensers. The tests showed that the mica in eight of the ten samples was suitable for used in condensers.

A Process for Utilizing Low-Grade Chrome Ores

Electrothermic fusion with lime increases the chromium-to-iron ratio and yields concentrates suitable for production of standard ferrochromium

AS A STEP in the direction of utilizing the low-grade chromium ores of the United States and Canada, the Chromium Mining & Smelting Corp., Sault Ste. Marie, Ont., Canada, has produced a high chromium-iron ratio concentrate, known as Chrom-X, suitable for the production of standard ferrochromium.

The process is described by Marvin J. Udy in the July, 1941, issue of *Metals and Alloys*, from which the following excerpt is taken:

United States and Canada have chrome ore deposits, but for the most part the ratio of chromium to iron is low and the standard grade of ferrochromium cannot be made from them. Any solution to the problem of using domestic low-ratio ores must be one that will produce at least a 68 to 70 percent ferrochromium or new products which are at least equivalent in cost and in performance to the standard ferrochromium. In our development we have fortunately been able to meet these requirements and have solved the problem in such a manner that we are able to produce concentrates of high ratio suitable for production of standard ferrochromium, or Chrom-X.

To produce concentrates of high ratio for the manufacture of ferrochromium a beneficiation is required to remove some of the iron and increase the chromium-to-iron ratio. This change of chromium-to-iron ratio is accomplished by fusing the ore or concentrate with lime and reducing the iron with part of the chromium, or by fusion of the ore with lime to replace the iron and part of the alumina from the chromite and then concentrating by water to remove the altered chromite of high chromium to iron ratio.

All chrome ores consist of a chromite and a silicate. The chromite is chiefly $FeO.Cr_2O_3$, with replacement of the FeO by MgO and the Cr_2O_3 by Al_2O_3. The silicate portion may vary considerably in composition. On the addition of lime it appears that MgO in the silicate is replaced, and in turn the MgO replaces FeO in the chromite. Increasing portions of lime then replace the FeO in the chromite and the Al_2O_3 and finally there is a complete decomposition of the chromite. If the quantity of lime is controlled so that complete decomposition is not reached, there results a constituent approaching magnesium chromite ($MgO.Cr_2O_3$), which crystallizes as an altered chromite of higher specific gravity than the original chromite of the ore and can be readily concentrated from the silicate portion by water concentration. This concentrate may have a chromium-to-iron ratio of 8 to 1 or more, and after acid treatment as much as 50 to 1. To produce ferrochromium of 70 percent grade this concentrate is simply used to sweeten a portion of the low-ratio ore to a 3 to 1 ratio or better, and then smelted by standard methods. The economic success of this method of concentration requires cheap power and other favorable economic factors.

From the standpoint of ferrochromium of standard grade, this method of concentration gives satisfactory and economical results. From the standpoint of Chrom-X, whether this concentration is necessary or desirable depends upon any given set of economic conditions. It is a method which can in some cases be used to advantage at the source of ore to save transportation charges and, carried out at such point, can produce high-grade concentrates in competition with the high-grade ores even in peace times.

To produce Chrom-X a direct reduction of the ore or concentrate is made in an electric furnace with coke to produce a chromium-iron metal of 8 to 10 percent C and up to 5 percent Si. High-carbon Chrom-X is made up of this metal mixed with sodium nitrate or other oxidizing agent such as sodium or calcium chromate in sufficient quantity to oxidize practically all of the silicon and part of the carbon. It is then briqueted and heated to bond it into a solid mass. The oxidizing agent forms the bond. Due to the fineness of the material and the thorough mixing, the product is uniform in composition.

Low-carbon Chrom-X is a compounded mixture of silicon or an alloy of silicon of very low carbon content (0.01 to 0.02 percent) and a mixture of the oxides of chromium and iron in chemical combination with lime. These oxides are made by oxidizing chromium-iron metal containing 8 to 10 percent C with lime at a high temperature and so controlling the operation as to get about 50 percent of the chromium as Cr_2O_3 and about 50 percent as CrO_3. These ratios may be varied to produce the correct amount of heat such that, when the Chrom-X reacts on the steel bath, it neither chills nor overheats the steel. The oxides and the silicon are ground together to minus 100 mesh and bagged for shipment or moistened and briqueted and heated to bond and form a solid brick. The silicon is present in about 10 percent excess over that required to reduce the oxides of iron and chromium. Carbon content is very low and will be of the order of 0.02 percent or less.

The company has reached a stage in its development far beyond the experimental. It has a plant in operation at Sault Ste. Marie, Ontario, with total capacity of 24,000 kw. and it contemplates the enlargement of this plant to meet the increased demands in Canada and England. The company further plans the building of a plant in the United States. Both plants will be able to operate on domestic or imported ores.

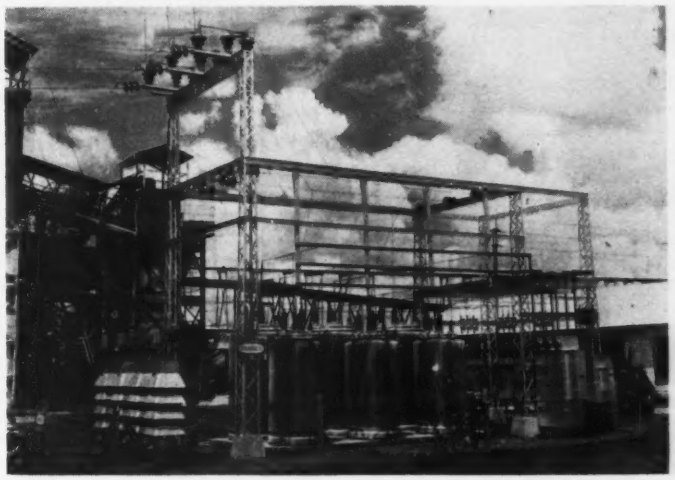
The principle applied in the manufacture of Chrom-X—namely, that of using the heat of combustion of silicon and carbon to melt metals and other materials—is capable of application to iron, manganese, silicon, vanadium, tungsten, nickel, copper, and other metals. It is the intention ultimately to produce all the alloys in form similar to Chrom-X. The whole scheme of metallurgy involved in the use of low-grade chrome ores and the resulting products and their uses have been the subject of a broad group of patent applications throughout the world, a number of which have been already granted and others will issue shortly.

The results of five years of research and endeavor to develop methods of using low-grade domestic ores or low-grade imported ores have developed an economically commercial process for producing (1) Chrom-X, (2) a concentrate from which Chrom-X or standard ferrochromium can be made, and (3) a process which is being expanded to the use of iron, manganese, silicon, vanadium, tungsten, nickel, copper, and the other alloying metals.

Southern Phosphate's Electric Load Now 6,500 Hp.

Including a new recovery plant, this Florida producer is using 200 electric motors ranging up to 750 hp. each

J. H. Edwards
Electrical Editor



Transformer capacity at Southern Phosphate's San Gully substation is 5,000 kva.

STRIP MINING, washing, concentrating, and drying of phosphate rock in the pebble field of Florida requires a large amount of electric power—on the order of 40 kw.hr. per ton of finished product. For the most part the larger operations of the district are 100 percent electrified. Through the courtesy of Chester A. Fulton, president of the Southern Phosphate Corp., one of the operating companies, and E. A. Pierce, vice president and manager, I am able to present here the results of an editorial survey of the electrical distribution and power applications at the company's mines and plants.

This company, which has its operating office at Bartow, is now producing at the rate of 600,000 tons per year. The output is derived from one mine known as the San Gully, which has a washer and a concentrator, and from a new reclamation plant, the Pauway, which works tailings from an earlier operation and replaces an old plant at Standard that was shut down last June. Concentrates from the two plants are dried at a third plant, at Ridgewood, where part of the output is also calcined. Common-carrier railroad haulage is used over the 9 miles from San Gully to Ridgewood and over the 20 miles from Pauway to Ridgewood. Save for this haulage all the other operations of mining, handling, concentrating, and drying are powered by electricity.

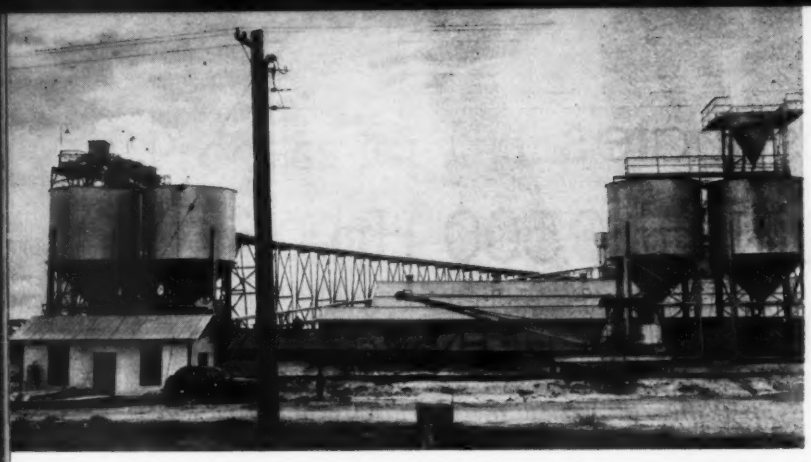
Mining consists of removing 15 to 20 ft. of soft overburden with a Bucyrus-Erie No. 320B electric dragline having a 138-ft. boom and a 9½-yd. alloy-steel bucket and then hydraulicking the 12-ft. bed of matrix with guns having 2¼-in. tips and operated at 200-lb. pressure. The matrix is pumped 1¼ miles to the plant. Power for the entire San Gully operation, including mining, transporting matrix, washing, and concentrating, is 39 kw.-hr. per ton. The drying and calcining plant at Ridgewood uses 1.33 kw.-hr. per ton.

Southern Phosphate purchases all its power at 33,000 v. from the Tampa Electric Co. The distribution lines carrying this voltage run from a central metering point at Mulberry and are owned and maintained by the power company, the total mileage to the active plants being 28.5. Three transformer substations reducing from 33,000 to 2,300 v. belong to the mining company which maintains them. A power factor of 90 percent is secured by using on the system three synchronous motors of 1,335 aggregate horsepower. In comparison, the connected horsepower of induction motors is 5,113. Non-inductive loads such as that of lighting are relatively small. In the concentrator at Pauway the illumination is by fluorescent units of high efficiency.

Alternating-current substations reducing from 33,000 to 2,300 v. at the three properties are in each case the outside type with steel structures and having G.E. transformers set on ground platforms. At San Gully a 5,000-kva. substation 150 ft. from the washer, 100 ft. from the hydraulic plant, and 1¼ miles from the pit consists of three 1,000-kva. and three 667-kva. transformers. At Pauway, a substation 1,000 ft. from the plant has three 500-kva. units. At the Ridgewood drying and calcining plant, a substation 600 ft. from that plant and within 150 ft. of the central machine shop and laboratory has three 200-kva. transformers for plant power and three 50-kva. for the laboratory and machine shop.

Lightning, according to Alexander (Monthly Weather Review, May, 1935), occurs more frequently in the area including Tampa and Lakeland than in any other section of the United States. In addition to the liberal use of arresters, a practice of the Southern Phosphate Corp. which is believed to have reduced lightning difficulties is the use of underground 2,300-v. feeders from transformer substations to plant loads. These are rubber-covered and braided cables in rigid galvanized steel conduit. Experience indicates that these underground runs in metal conduit choke off to some extent the transmission of lightning surges to plant motors. Safety and appearance are also improved by these conduit entrances to the plants.

At the San Gully substation, even the No. 4/0 2,300-v. line feeding the mine is underground for the first 225 ft., which is across the highway to the beginning of a pole line. Beyond a point where No. 1 booster pump is fed, the pole-line wire size is dropped to 1/0. To operate the dragline, which is



From the pole line in the foreground the 2,300-v. power enters Pauway plant via an underground conduit. The substation stepping down from 33,000 to 2,300 v. is 1,000 ft. distant

caterpillar mounted and has Ward Leonard control with a 435-kva. 0.8-power-factor synchronous motor, a trailing cable circuit 1,250 ft. long is attached directly to the end of the pole line with Kearney hot-wire clips. This trailing circuit consists of three single-conductor No. 1/0 Rome-General cables, rubber-insulated for 5,000 v. and covered with hard loom jacketing. The cables are paid out from a manually wound reel mounted under the main frame of the dragline. Rollers on a cross member serve to guide the cables and protect them from wear.

Secondaries of the 33,000- to 2,300-v. transformers supplying the dragline, pumps and plants are connected delta. Thus the distribution is operated without a power ground connection and without ground conductors being included in the pole lines and trailing cable. Electricians take action immediately to clear a conductor or machine of a ground if one appears. This contingency is indicated by the dimming or going out of a lamp and the brightening of two others.

These ground-detecting lamps are mounted in plain sight at the substation and are connected separately across the secondaries of instrument transformers the primaries of which are connected Y and have their common or neutral point grounded (for further details see p. 63).

The pit pump and two boosters which transport matrix 1¼ miles to the washer are duplicates. Pumps are 12-in. heavy-duty sand type made by the Georgia Iron Works and the motors General Electric 300-hp. 2,300-v. with rotor-resistance speed control. At San Gully several 150-hp. pumping units are engaged in reclaiming old tailings.

A Peerless turbine pump in a deep-well used when rains fail to supply sufficient surface water for the San Gully operations is driven by a 100-hp. General Electric Type FT motor

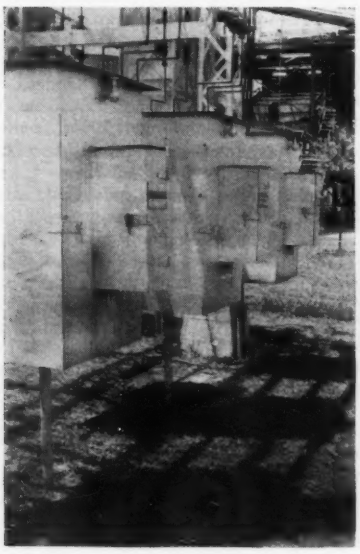
Electric Motor Data, Southern Phosphate Corp.

(Fractional Horsepower Sizes Not Included)

	All Types		100-Hp. and Larger Sizes		2,300-V. (Note 1)		Synchronous		Wound Rotor	
San Gully mine and plant	No.	Conn. Hp.	No.	Conn. Hp.	No.	Conn. Hp.	No.	Conn. Hp.	No.	Conn. Hp.
Dragline	1	435	1	435	1	435	1	435	0	...
Hydraulic station	5	1,200	2	900	5	1,200	1	750	0	...
Pit and booster pumps	3	900	3	900	3	900	0	...	3	900
Reclamation pumps	5	675		600	5	675	0	...	4	600
Deepwell pump	1	100	1	100	1	100	0	...	0	...
Washer	26	1,066	3	625	8	920	0	...	0	...
Concentrator	53	205	2	200	3	230	0	...	0	...
Pauway concentrator	51	1,264	5	1,025	10	1,155	1	150	3	500
Ridgewood drying plant	37	513	0	....	6	280	0	66	2	120
Shops and laboratory	18	90	0	....	0	....	0	...	0	...
Grand totals	200	6,448	20	4,785	42	5,895	3	1,335	12	2,120

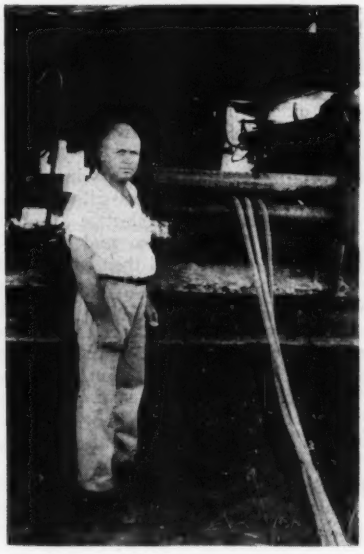
Note 1: All other motors operate on 220 v.

Conduits entering the ground from three FK35 overload circuit breakers at San Gully substation feed the dragline, pit, hydraulic station and concentrator



Dragline trailing cables attached directly to the line with hot wire clamps supply power to a Bucyrus-Erie No. 320B, 9½-cu.yd. electric dragline

Rollers for single-conductor trailing cables on the dragline. Back of W. W. Faulkner, chief electrician, hangs an insulated hook for moving cables



started by a CR-1034 manual compensator.

Water at 200-lb. pressure for three hydraulic giants which mine matrix is pumped by one hydraulic station close to the washer and concentrator. It is supplied by one pumping unit made by Allis-Chalmers and consisting of a 5,000-g.p.m. 475-ft. head centrifugal pump driven by a 750-hp. 2,300-v. synchronous motor designed for unity power factor. The third synchronous motor, one which is rated 150 hp. 0.8 power factor, is at Pauway reclamation plant and it likewise was made by Allis-Chalmers and drives a 2,000-g.p.m. pump for mining the tailings.

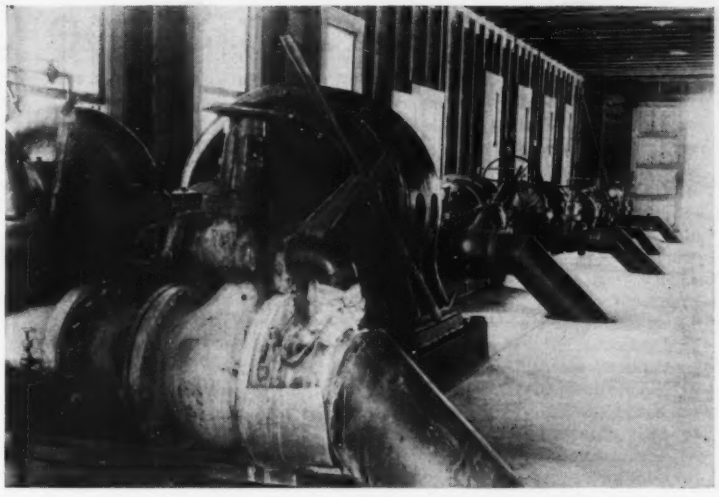
From the accompanying table giving numbers of motors and total connected horsepower it will be observed that of the 200 motors 42 are operated at 2,300 v. but that the combined horsepower of the 42 is 5,895, which is 93 percent of the total connected load.

Motor sizes average 32¼ hp. and the 20 motors of 100 hp. or above total 4,785 hp. A larger proportion of the motors are ball-bearing at the Ridgewood plant than at the others: 17 out of 37. At Pauway the following motors have ball bearings: one 125-hp., one 100-hp., two 50-, one 30-, two 10-, one 7½- and two 5-hp. For the most part General Electric motors are used at all operations. At San Gully, for instance, all of the 88 motors are of that make excepting that four of the five in the hydraulic plant are Allis-Chalmers.

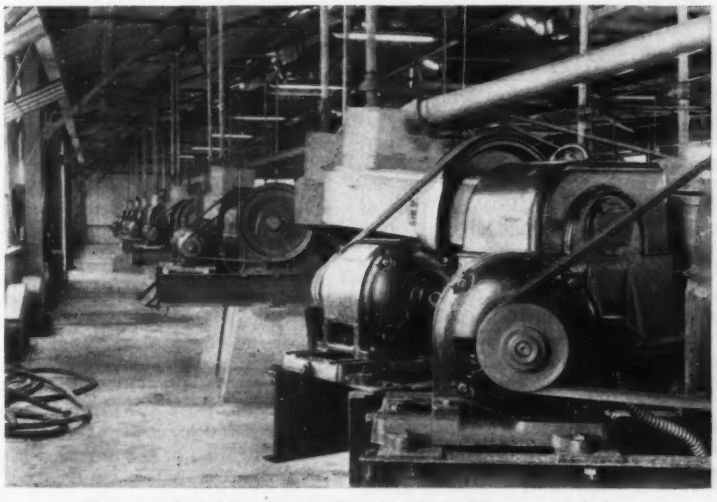
Excepting a limited amount of "Busduct," all wiring runs in the plants are protected in rigid, steel galvanized conduit. The wires in the conduits carrying 2,300-v. power are rubber-insulated for 5,000 v. and have single braid. As a rule, wires larger than Underwriter's code requirements are used; also, the conduits are the next size larger than the minimum called for by the code.

The "Busduct," made by Frank Adam Electric Co., is used in the Pauway reclamation plant which was completed in June, 1941. This square steel duct encloses 220-v. buses supplying power to the smaller loads, including 36 G.E. 2-hp. motors driving Superduty diagonal-deck No. 6 concentrating tables. General lighting in the table building consists of 30 Mitchell fluorescent units each containing two General Electric 40-watt tubes. In addition to them, six 200-watt Mazda lamps are in use, principally in wet places in the plant.

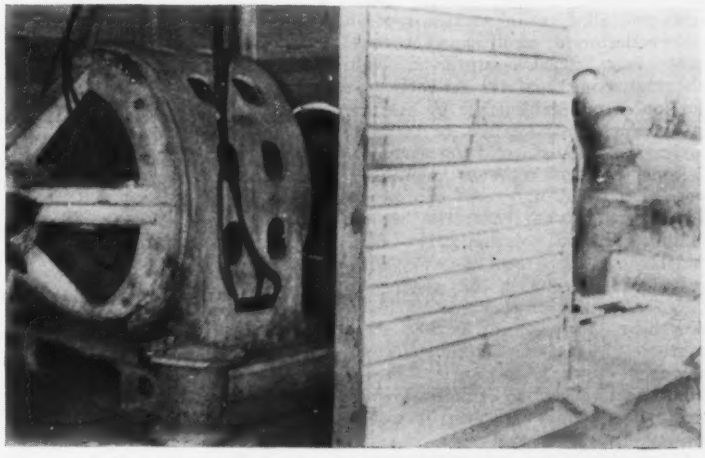
Electrical construction and maintenance is supervised by W. W. Faulkner, chief electrician, who has headquarters at a central electrical repair shop at San Gully. Electrical winding is done there and on the recent jobs glass-insulated coils have been applied. The 300-hp. 2,300-v. motor driving No. 1 booster pump is also operating with that type of insulation.



San Gully hydraulic station. Pump in foreground is driven by a 750-hp. synchronous motor



Sixteen of the table motors in Pauway plant. Motors are fed from an overhead bus duct. Lighting is by 30 fluorescent fixtures



Glass-insulated 300-hp. motor with rotor-resistance speed control driving No. 1 booster pump at San Gully pit

Recovering Slime Cassiterite by Soap Flotation

Commercial possibilities of this procedure are checked at Pirquitas Plant, in Argentina

Ernest Oberbillig*
Yellow Pine, via Cascade
Idaho

E. P. Frink
Resident Manager, Soc. Min. Pirquitas, Picchetti y Cia., Jujuy, Argentina

CASSITERITE ORES from lode deposits have long been concentrated by gravity methods. Stage crushing and grinding followed by gravity concentration at its best will yield about 70 percent recovery. The other 30 percent or more is lost in the tailings, mainly in the form of fines or "slimes." Bolivia alone has millions of tons of tailings which average about 1 percent tin. These rich dumps lie awaiting the development of a flotation process for extraction of the cassiterite and could be a valuable source of tin.

Flotation is being applied to tin ores by floating detrimental sulphide minerals (mostly pyrite and arsenopyrite) away from cassiterite in a gravity concentrate. This operation is successful inasmuch as the non-float can be cleaned to as low as 1 percent sulphur with a recovery of 95 percent or better of the tin in this product. However, direct flotation of cassiterite from its ores has not advanced much beyond the laboratory stage. Besides fatty acids and alkali metal oleates, several other collectors or conditioning schemes have been tried: cupferron and phenylglyoxine, coal tar products, reduction of mineral surface by heating in a reducing atmosphere, or metallic coating on the surface of the particles.

None of the reagents mentioned were checked by us except cupferron. We floated pure cassiterite in a sodium hydroxide solution (pH = 8.8) and made a 72 percent recovery with an alcoholic solution of cupferron (1.5 kg. cupferron per metric ton). We have also found Z=10 hexyl xanthate to have a noticeable collective tendency toward cassiterite in a slightly acid circuit. The "metallizing" or "gas-reduction" schemes, it would appear, are rather costly.

Cassiterite has been found floatable by oleic acid, or modified oleic acid. The difficulty has been that associated gangue minerals, such as hematite and less often barite, collected in the concentrate. Repeated cleaning does not give a marketable concentrate from most ores. It can be assumed that tabling would give some results on a flotation concentrate containing mostly hematite and cassiterite, since Handy and Beard[1] used a table for producing a marketable tin concentrate from a low-grade flotation product.

Electromagnetic separation without roasting has been tried for separating hematite from cassiterite. Results obtained indicate a clean separation of the two minerals contained in a buddle concentrate similar to the one used later in this work. The cassiterite was all minus 200 mesh.

Investigation of the commercial possibilities of applying soap flotation for recovering "slime" cassiterite is here reported. It was conducted as follows:

1. Flotation studies of pure cassiterite. 2. A hematite-cassiterite mixture was studied to the end of separating the two minerals by flotation. 3. A natural plant tailing from an oxide ore was treated by flotation followed by tabling of the flotation concentrate, containing mostly hematite and cassiterite. This study was made in the laboratory of Soc. Min. Pirquitas Picchetti y Cia., Mina Pirquitas, Jujuy, Argentina.

Material Used.—The cassiterite-bearing materials used were: a pure cassiterite, a natural cassiterite-hematite buddle concentrate, and a flotation-plant tailing.

The first was a placer concentrate—the cleanest mineral obtainable here. Table I gives its chemical and screen analysis. It was dry-pulverized to pass a 100-mesh Tyler sieve and stored in stoppered glass bottles.

The natural cassiterite-hematite buddle concentrate was collected by buddling current oxide-ore tailings and taking only the minus-200-mesh concentrate. Its analysis is given in Table II. This concentrate mineralogically approximated a flotation concentrate made from the plant tailings except that it contained no slimes.

The flotation tailings studied were the accumulation of three years' work on a rich silver-tin ore in its oxide form. About 80 percent recovery of silver and 50 percent recovery of tin were made. Tin and iron analysis (calculated) is given in Table V and Table VI. This was a natural tailing containing a high percentage of slimes.

Laboratory Methods—A small, glass pneumatic flotation cell was used to study pure cassiterite with various reagents. This cell has been described in the *Mining Journal*, May 30, 1938. A 25-gm. sample was taken for each test. Conditioning took 2½ minutes and flotation 5 minutes. The concentrate was collected, filtered on a tared filter paper, and weighed. Percent recovery was calculated from the weight of concentrate and weight of initial test feed. Test of pH was made on the concentrate filtrate with a La Motte block comparator.

For testing the buddle concentrate and plant tails 500- and 2,000-gm. Denver (Fahrenwald) laboratory flotation machines were used.

Commercial grades of sodium hexametaphosphate (supplied through the courtesy of W. E. Keck), DuPont B-23 frother, and tannin were used; the rest of the reagents were chemically pure. Also, distilled water was used. All reagents were added in a water solution containing 10.0, 1.0, 0.1,

* At the time of writing this article the senior author was at Mina Pirquitas, in Argentina.

[1] R. S. Handy and R. R. Beard. "Concentrating Cassiterite Ores," U. S. Patent No. 1737717. Dec. 3, 1930.

or 0.01 mg. of reagent per cubic centimeter. Then, 5, 10, or 25 cc. of reagent solution was added, using a pipette. Reagents are shown in concentrations of the anhydrous state. Metric weights were used.

1. Experimental data on pure cassiterite—Collectors. Work with the pure cassiterite was begun by studying three collectors—oleic acid, palmitic acid, and sodium oleate. Emulsol X-1 was first tried as an emulsifier for the two fatty acids, but its use was stopped because of excessive frothing in higher concentrations of the collector. Use of X-1 alone did not collect the cassiterite. Then the work was continued using triethylamine as an emulsifier for the oleic and palmitic acid at 10 percent concentration of the emulsifier. Sodium oleate was made into a water solution, using warm distilled water and diluting to definite concentrations. In normal concentrations of the collectors, an auxiliary frother was required and DuPont B-23 was selected. DuPont B-23, also, is a non-collecting frother for cassiterite in soap flotation.

Fig. 1 shows recoveries at various concentrations of the three collectors. Emulsified oleic acid gave the best recovery, and it was decided to use it as the collector at 0.40 kg. per metric ton in the rest of the test work on pure cassiterite. The concentrate filtrate was approximately neutral.

At higher concentrations of the collector where the recovery was good the non-float cassiterite was the coarse particles rather than the slime, being a clean cassiterite sand 100 to 200 mesh in size.

Acids and Bases—Sulphuric acid and phosphoric acid were studied as well as the two bases sodium hydroxide and sodium carbonate. Fig. 2 shows the cassiterite recovery at various concentrations as well as approximate pH values. Both acids showed powerful depressing qualities, possibly due to decomposition of oleic acid by the mineral acids. The two bases might be expected to give similar results because of the basic reaction of sodium carbonate:

$$Na_2CO_3 + 2HOH = 2NaOH + H_2CO_3$$

This reaction proceeds to the right, and the weak carbonic acid loses in the competition against the strong base and the reaction continues to a pH of about 9.6, where it is "buffered." In the lower concentrations, the bases are not such strong depressants, but at higher concentrations they are.

It has been noted from sulphide flotation practice with a pulp containing cassiterite that a lime alkaline circuit will "froth-float" approximately twice as much tin as is collected in an acid

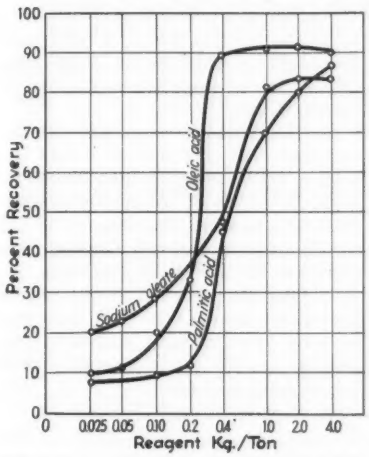


Fig. 1 . . . Recovery of pure cassiterite with three collectors. pH=6.9-7.0. B-23 reagent, 0.15–0.25 kg./metric ton

Table I—Screen and Chemical Analysis of Pure Cassiterite Used in Investigation

Product	Wt.	% Wt.	Assay % Sn	Assay % Fe	% Wt. x Assay Sn	% Wt. x Assay Fe	% Distribution Sn	% Distribution Fe
−100/+150 mesh...	20.3	11.40	71.15	1.15	8.11	0.13	11.51	9.50
−150/+200 mesh...	21.7	12.19	70.85	1.40	8.64	0.17	12.28	12.40
−200 mesh.....	136.0	76.41	70.15	1.40	53.60	1.07	76.21	78.10
Totals.....	178.0	100.00	.....	....	70.35	1.37	100.00	100.00

Table II—Chemical Analysis of 200-Mesh Buddle Concentrate

Sn %	Fe %	Insol. %
31.28	26.15	17.46

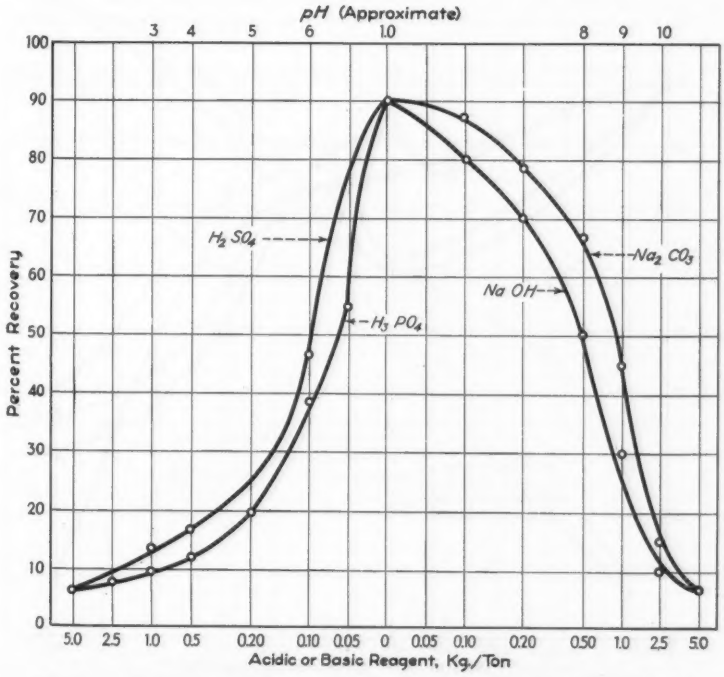

pH (Approximate)
3 4 5 6 1.0 8 9 10
Percent Recovery
Acidic or Basic Reagent, Kg./Ton
H2SO4
H3PO4
NaOH
Na2CO3
5.0 2.5 1.0 0.5 0.20 0.10 0.05 0 0.05 0.10 0.20 0.50 1.0 2.5 5.0

Fig. 2 . . . Recovery of pure cassiterite with acids and bases. Constant 0.40 kg. oleic acid per metric ton and 0.15–0.25 kg. B-23 reagent per metric ton

Table III—Recovery of Pure Cassiterite Using 2 kg./ton $FeCl_3$ and Variable Quantities of Oleic Acid: pH = 6.9 (After Flotation)

Used constant 0.15 K/T DuPont B-23 frother.

Kg./Ton Oleic Acid	Recovery, %
0.40	30.0
0.80	38.8
1.20	45.6
1.60	60.5
2.00	74.9

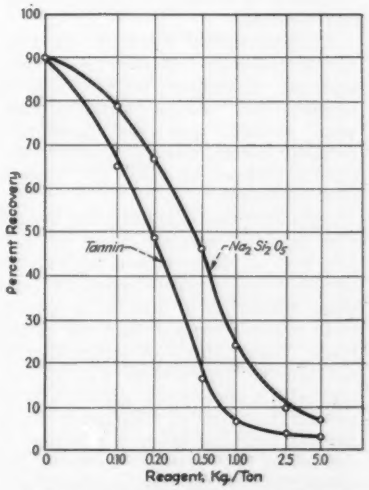


Fig. 3 . . . Flotation of pure cassiterite with colloidal reagents, 0.40 kg. oleic acid per metric ton and 0.15–0.25 kg. B-23 frother per metric ton. pH=6.9-7.0

Table IV—Flotation Separation of Hematite and Cassiterite

Condition 2½ minutes; float 5 minutes, with 200 grams of mixture — buddle concentrate. (−200 mesh.) Reagents; Oleic acid, 0.20 kg./ton. DuPont B-23 frother, 0.15 kg./ton. $FeCl_3$, $Al(NO_3)_3$, $Na_2Si_2O_5$, and $Na_4P_6O_{18}$ as shown below. pH = 6.9−7.1.

Test	Kg./Ton Reagent Used	Recovery of Sn in Concentrate, %	Rejection of Fe in Tailing, %	% Sn by Assay in Concentrate
	1.00	96.5	4.5	31.45
	1.10	56.0	60.5	38.35
a	1.20	31.0	77.8	37.40
$FeCl_3$	1.30	18.9	86.3	37.85
	1.40	17.0	88.2	37.00
	1.50	5.6	95.5	34.80
	1.00	78.8	37.1	35.95
	1.10	67.4	49.4	34.65
b	1.20	31.2	76.8	35.55
$Al(NO_3)_3$	1.30	21.9	85.8	38.40
	1.40	11.5	91.5	36.40
	1.50	7.9	94.1	35.95

Test	Kg./Ton Reagent Used	Recovery of Sn in Concentrate, %	Rejection of Fe in Tailing, %	% Sn by Assay in Concentrate
	0.10	87.5	30.5	36.00
	0.20	88.0	30.8	36.10
c	0.30	89.0	32.5	36.65
$Na_2Si_2O_5$	0.40	57.2	95.6	43.65
	0.50	42.5	95.1	43.50
	0.60	7.3	95.8	43.60
	0.05	85.0	37.0	32.20
	0.10	82.0	46.0	38.85
d	0.15	66.0	61.2	40.50
$Na_4P_6O_{18}$	0.20	35.0	85.0	45.55
	0.25	25.4	89.9	45.65

(usually sulphuric) circuit. However, cassiterite thus floated in this manner is usually negligible.

Colloidal Reagents—Sodium silicate and tannin were found to be depressants of pure cassiterite at higher concentrations. Fig. 3 shows the results. The pH of the concentrate filtrate was approximately neutral.

Metallic Salts — Fig. 4 shows results obtained with lead acetate, copper acetate, ferric chloride, and aluminum nitrate. Depression at higher concentrations of metallic salts could be due to "salting out" of the collector oleate ions by the metallic ions. This is shown in Table III by an increase in recovery at higher oleic acid concentrations when using a constant 2 kg. of ferric chloride per ton.

Phosphates—Three sodium phosphates—meta-, hexameta-, and monobasic—were studied as a final set of reagents with pure cassiterite. Results are shown in Fig. 5. The data show that the three phosphates alone with emulsified oleic acid are pure cassiterite depressants. Sodium metaphosphate was selected for further study with the hematite-cassiterite mixture to attempt a flotation separation of the two minerals.

2. Experimental Data on the Cassiterite-Hematite Mixture — Ferric chloride, aluminum nitrate, sodium silicate, and sodium metaphosphate were studied with the cassiterite-hematite natural mineral mixture. Results are shown in Table IV, Tests *a* to *d* inclusive.

Metallic Salts—Flotation test results with ferric chloride are shown in Table IV, Test *a*. This reagent appears to depress the hematite partially and cause cassiterite to float as a concentrate richer in grade than the initial feed. With 1.10 kg. of ferric chloride per ton the tin recovery was 56 percent with a concentrate grade of 38.35 percent tin (initial feed grade 31.28 percent tin) and an iron rejection in the tail of 60.5 percent. However, the separation of the two minerals was relatively poor. Also, most metallic salts must, usually, be carefully excluded from a soap flotation circuit containing silicate minerals. The latter would be activated and floated.

Aluminum nitrate gave a poorer separation than ferric chloride of the two minerals, shown by Table IV, Test *b*.

Sodium Silicate—It was pointed out in the section on the study of pure cassiterite that sodium silicate will effectively depress quartz (and presumably most other silicates) when floating cassiterite. Table IV, Test *c*, shows that it will also cause a fair separation of hematite and cassiterite. At 0.4 kg. of sodium silicate per ton on mineral mixture, the concentrate assayed 43.65 percent tin at a recovery of 57.2 percent; the iron distribution in the tail was 95.6 percent. Results have shown that carefully controlled quantities will depress quartz and make a fair separation of hematite when floating cassiterite. Therefore it should be considered for commercial application of soap flotation to cassiterite recovery.

Sodium Metaphosphate—It was anticipated that this reagent would exhibit a better behavior than shown by Table IV, Test *d*. At higher reagent concentrations an enrichment of tin in the concentrate and likewise iron rejection in the tail was noticed, but at a poor tin recovery. These results are exactly opposite to those indicated by the pure mineral studies.[2] This reagent was the last to be studied with the mineral mixture.

Laboratory Testing of Plant Tails—Two of the many tests done on the plant tailing are reported. These are both representative for each treatment. The first test, shown in Table

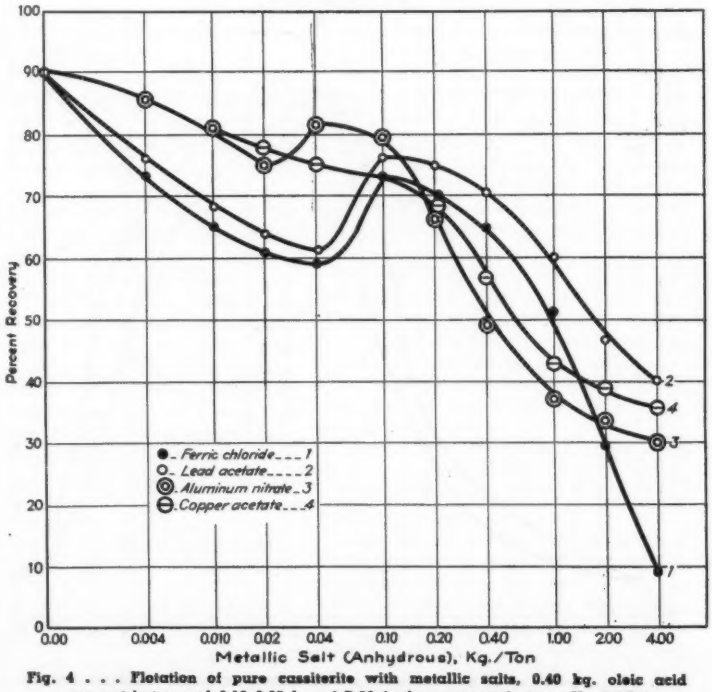


Fig. 4 . . . Flotation of pure cassiterite with metallic salts, 0.40 kg. oleic acid per metric ton and 0.15–0.25 kg. of B-23 frother per metric ton. pH=6.9-7.0

V, gives results of direct flotation and flotation cleaning of the rougher concentrate. Table VI shows results of table cleaning of the flotation rougher concentrate.

Flotation cleaning of a low-grade rougher flotation concentrate appears to give a low recovery of the cassiterite in a medium-grade concentrate; too much tin remained in the middling or cleaner tailing.

Table cleaning of a low-grade rougher flotation concentrate gave almost an 80 percent recovery of the tin and an over-all recovery of 65 percent; the concentrate grade was 45 percent. Since the tin recovered by careful tabling was all minus 200 mesh, this would indicate that good classification followed by slime-table operation will effect a fair tin recovery. Hematite floated in the rougher concentrate was mostly coarse particles, the cassiterite being very fine.

Table V—Flotation of Plant Tailing and Flotation Cleaning of the Rougher Concentrate

Material, −65 mesh plant tailing. Reagents, rougher (10 minutes); $Na_2Si_2O_5$, 0.22 kg./ton; oleic acid, 0.26 kg./ton; B-23, 0.06 kg./ton. Cleaner (10 minutes); $Na_2Si_2O_5$, 0.15 kg./ton; B-23, 0.02 kg./ton. pH (rougher and cleaner) = 6.9.

Product	Wt.	Assay			% Wt. x Assay		% Distribution	
	Grams	% Wt.	% Sn	% Fe	Sn	Fe	Sn	Fe
Concentrate	162.5	8.56	31.35	26.50	2.684	2.269	43.38	16.72
Middling	528.5	27.85	8.63	17.75	2.403	4.942	38.84	36.42
Tailing	1,207.0	63.59	1.73	10.00	1.100	6.359	17.78	46.86
Total (heads)	1,898.0	100.00	6.19	13.57	6.187	13.570	100.00	100.00

Table VI—Flotation of Plant Tailing and Table Cleaning of Rougher Concentrate

Material, −65 mesh plant tailing. Reagents, flotation (15 minutes); $Na_2Si_2O_5$, 0.34 kg./ton; Oleic acid, 0.26 kg./ton; B-23, 0.08 kg./ton. Table. No reagents used for dispersion. pH = 7.0.

Product	Wt.	Assay		% Wt. x Assay		% Distribution	
	Grams	% Wt.	% Sn	% Fe	Sn	Fe	Sn	Fe
Table concentrate*	56	2.35	44.95	12.90	1.0563	0.3032	64.99	3.08
Table middling	225	9.45	2.35	7.65	0.2221	0.7229	13.67	7.35
Table tail	76	3.19	1.55	6.65	0.0494	0.2121	3.04	2.16
Total flotation concentrate	357	14.99	8.86	8.26	1.3278	1.2382	81.70	12.59
Flotation tail	2,024	85.01	0.35	10.12	0.2975	8.6030	18.30	87.41
Total (heads)	2,381	100.00	1.63	9.84	1.6253	9.8412	100.00	100.00

* Table concentrate is all minus 200 mesh.

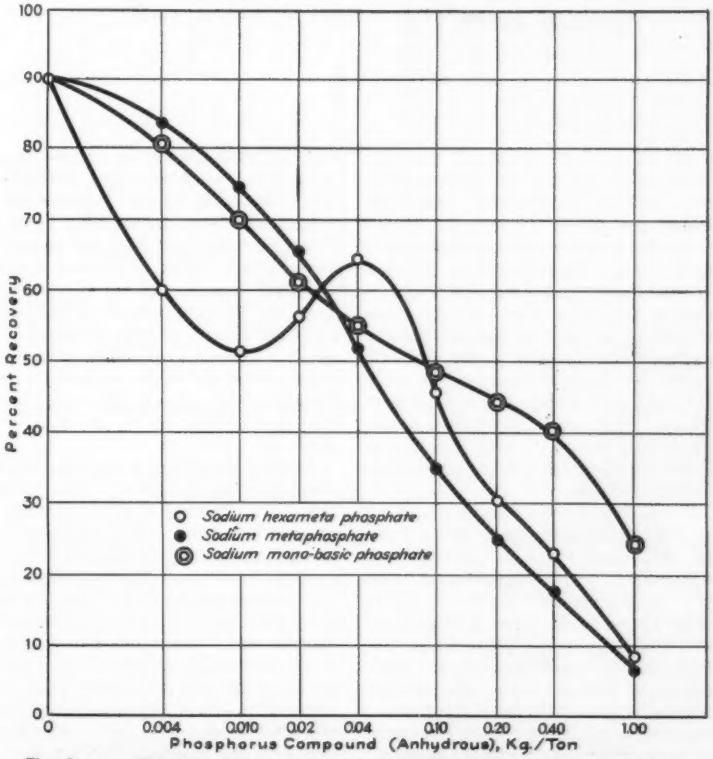


Fig. 5 . . . Flotation of pure cassiterite with phosphorus compounds, 0.40 kg. oleic acid per metric ton and 0.15–0.25 kg. B-23 reagent per metric ton. pH = 6.9-7.0

This testing indicates that flotation alone with the common soap reagents will not yield a tin recovery and corresponding concentrate grade worth commercial consideration on the ore studied. However, rougher flotation followed by some gravity "clean-up" method on the flotation concentrate should yield a recovery of about 60 percent and an easily marketable concentrate of at least 40 percent tin grade. Either careful table or tina operation should be tried. Also, it was pointed out earlier that an electromagnetic test gave good separation of hematite and cassiterite in a mineral mixture similar to a flotation rougher concentrate. Therefore, electromagnetic separation should also be tried; at least, a test should be made on a representative cassiterite flotation concentrate.

Conclusions

1. Of the three collectors studied, emulsified oleic acid showed the most powerful collecting properties for cassiterite at a neutral pH.

2. The two mineral acids studied were powerful cassiterite depressants.

3. The bases, sodium hydroxide and carbonate, showed depressing qualities in higher concentrations, but at lower reagent concentrations were not such powerful depressants.

4. Sodium silicate and tannin are powerful depressants for cassiterite, but sodium silicate in a circuit containing silicate minerals and hematite will preferentially depress these minerals and allow a fair flotation of the cassiterite.

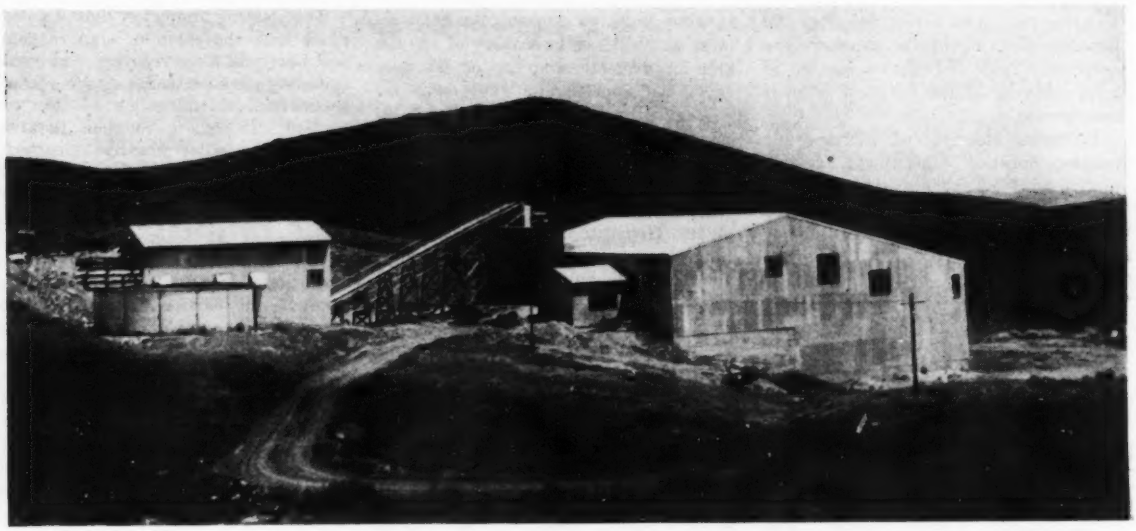
5. Metallic salts show depressing qualities for cassiterite at higher concentrations, but at lower concentrations, ferric chloride, lead acetate, and aluminum nitrate show a depression, and at slightly higher concentrations increased flotation of cassiterite; then a recovery decline on further increase of metallic reagents.

6. The three phosphate compounds act as cassiterite depressants. Sodium hexametaphosphate showed a "dip" in the recovery curve similar to the metallic salts.

7. With a natural ore containing only cassiterite and untarnished quartz, sodium silicate and oleic acid with one of the alcohol frothers should give a clean flotation separation of the two minerals.

8. With a natural ore containing mostly silicate minerals, hematite and cassiterite, sodium silicate with oleic acid, and an alcohol frother will give about an 80 percent tin recovery with a concentrate low in tin grade containing mostly cassiterite and hematite. This low-grade concentrate can be "graded up" to a marketable tin concentrate by careful gravity methods or electromagnetic separation.

[2] W. E. Keck, G. C. Eggleston and W. W. Lowry: "Study of the Flotative Properties of Hematite." *Trans.*, A.I.M.E. 1939.



Crushing plant and 250-ton cyanide mill at the Adelaide Crown

An Open-Pit Gold Mine in Nevada

The Adelaide Crown mines and cyanides 250 tons of ore per day with 18 men

John B. Huttl
Assistant Editor

THE ADELAIDE CROWN MINE, which I visited during a recent trip to Nevada and Utah, is in the Adelaide district of Humboldt County, Nev., at about 6,000 ft. elevation. It is easily reached by car over a gravel road that branches off U. S. Highway 40 at Golconda, the nearest railroad point, 12 miles north. The property has been developed intermittently by several owners in the past, the most work having been done from 1922 to 1924 while it was under option to the W. C. Hammon interests, of San Francisco. Adelaide Crown Mines, a company headed by Roy A. Hardy, consulting engineer of Reno, entered the district in 1939 and completed a 250-ton cyanide mill early in September, 1940.

Current operations are centered at an open pit 500 ft. long on a lens averaging 50 ft. in width and outcropping for a mile. The lens consists of broken-up quartz in shale, containing sugar quartz and streaks of manganese, and assaying about $5.50 per ton. At one point underground operations are to be started soon, with ore to be stoped by underhand methods. Although the mining and milling practice follows established standards closely, the mine is of interest because of the tonnage that is handled with a small force. Approximately 18 men are employed, including the superintendent and office workers. Of these, two are at the pit, two per shift at the mill, two on one shift at the crushing plant, one clean-up man on day shift only, and repair men and roustabouts. Ore is loaded and transported under contract by Dodge Construction Co., of Fallon, Nev. Current daily production is about 250 tons.

At the pit the two company men do the drilling and blasting. Blast holes are put down with a conventional wagon drill, and blockholing is done with jackhammers. Rods of 1⅛-in. hollow-round steel fitted with detachable bits are used. Blast holes are drilled 20 ft. apart and 20 ft. deep, staggered hexagonally. The charge consists of bag powder of 40 percent strength, about 150 lb. being consumed per hole. Springing and blockholing are done with 45 percent dynamite and electric caps. Holes in tough ore are sprung up to three times, using 8 to 30 sticks each time. Broken ore is loaded into 8-ton trucks by a 1¼-cu.yd. Northwest diesel shovel, and hauled to the crushing plant, a distance averaging half a mile. Material stockpiled near the dumping platform is pushed over the grizzly at intervals by a bulldozer. The oxidized ore milled contains silver and gold principally in a ratio of about 30 to 1. The metal particles are so fine as to be nearly invisible.

Milling procedure is explained in the accompanying flowsheet. The grizzly has 12-in. spacing and is provided at the bottom with a small platform for storing and subsequently reducing the oversize boulders. The undersize falls into one of two sections (one for custom ore) of the 500-ton coarse-ore storage bin. Coarse material goes by a 30-in. belt conveyor to a 3-ft. Traylor gyratory set at ⅝-in.; the fines are conveyed to a 4x8-ft. Symons double-deck vibrating screen working in closed circuit with the crusher. The top deck is covered with a screen having 2-in.

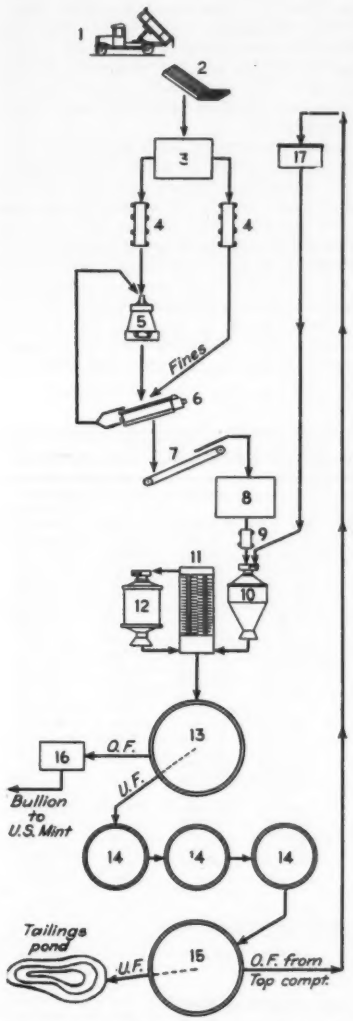
openings, and the lower deck with screen having ⅜-in. holes. Minus ⅜-in. material is conveyed by a 24-in. incline belt to the 850-ton fine-ore bin. From here a conveyor feeder directs it to a 7x8½-ft. P. & M. ball mill in closed circuit with an 8-ft. rake classifier, grinding to 5 to 10 percent plus 65 mesh.

All milling is done in cyanide solution. Classifier overflow is delivered by a 3-in. Wilfley pump to a 45x12-ft. primary thickener, the overflow of which is sent to Merrill-Crowe precipitation. Underflow goes successively to three 29½x29½-ft. agitators, and a four-tray, 45x30-ft., Dorr washing thickener. The agitation period in each tank is about 30 hr. Consumption of lime and cyanide per ton of ore treated is 5 lb. and 0.6 lb. respectively, each being added at No. 1 agitator.

Save for adding zinc dust to the press to avoid precipitation of silver in the pipe lines, which is done by a small triplex pressure pump below the zinc cone, precipitation operations follow standard practice. Zinc consumption is about 0.19 lb. per ton. Recovery of gold is 90 percent and that of silver 65 percent. Material betterment of these figures is expected after a primary Hardinge ball mill has been installed. The ore will then be ground to pass a 100-mesh screen.

At the refinery two oil-fired Monarch barrel furnaces are available for smelting. Flux used averages 25 percent soda, 30 percent borax, 15 percent silica, 5 percent niter, and 10 percent hematite. The resulting bullion, 985 fine, is shipped to the San Francisco Mint.

Water for camp and mill is obtained from old underground workings, where a 350-g.p.m. Byron-Jackson centrifugal pump is installed. Electric power is brought to the property by a 60,000-v. line of the Sierra Pacific Power Co. This voltage is stepped down to 440 v. in a substation near the mill. J. C. Shoemaker, superintendent, is in charge.



Flowsheet of Adelaide Crown cyanide plant

LEGEND

1. 8-ton trucks
2. Steel grizzly
3. 250-ton coarse-ore bin
4. 30-in. belt conveyors
5. 3-ft. Traylor gyratory crusher
6. 4x8-ft. Symons double-deck vibrating screen
7. 24-in. belt conveyor
8. 850-ton fine-ore bin
9. Conveyor feeder
10. Hardinge ball mill (in course of installation)
11. 8-ft. Dorr classifier
12. 7x8¼-ft. P&M ball mill
13. 45x12-ft. primary thickener
14. 29½x29½-ft. agitators
15. 45x30-ft., four-compartment Dorr washing thickener
16. Precipitation plant
17. Mill tank

Starting to strip at a new point at the Adelaide Crown. The 1¼-cu.yd. diesel shovel is loading an 8-ton truck. Main pit in background

Bulldozer pushing ore stocked on dumping platform over grizzly above storage bin

Summary of the

MARKETS

WITH price changes in non-ferrous metals few and far between, even minor revisions are now worthy of special comment. During November, both silver and quicksilver were raised in price. To satisfy the call for silver for industrial purposes, the leading interest suddenly found himself in direct competition with the Government in obtaining sufficient supplies of foreign metal. Under an agreement between Mexico and the United States, the Treasury again will purchase Mexico's surplus of newly mined silver on the basis of 35¢ an ounce. This action, which took place late in the month, forced the New York open market quotation for silver of foreign origin to 35⅛¢. Quicksilver is being produced in this country at the highest rate on record for modern times, yet production appears to be sold up and the market continues to gain strength, the price moving up $2 per flask to $197 minimum, New York.

Both copper and zinc presented nothing new pricewise. The trade is wondering whether Metals Reserve will have to pay more for Chilean copper when the deal comes up for renewal around the turn of the year. Domestic apparent consumption of copper was 137,000 tons during October, which compares with 134,000 tons in September. The November rate of disappearance was up to the average of recent months. Domestic production of refined copper has been holding around 85,000 tons a month, with imports totaling more than 60,000 tons a month. Sales of domestic copper for November amounted to 78,577 tons.

The emergency "pool" for zinc for December was lowered 2 percent to 29 percent of the amount produced during August. The plan under which zinc is being merchandised is said to be working smoothly.

After informing lead producers that the present price of 5.85¢ was adequate for enlarging output, thereby almost killing all thought of a higher price, the price question was revived by a House committee and the subject comes up for review Jan. 5. Domestic producers have been ordered to set aside 15 percent of output for an emergency supply.

The *E. & M. J.* index of non-ferrous metal prices for November was 85.32, against 83.84 in October and 83.14 a year ago.

UNITED STATES MARKET

1941 Nov.	Electrolytic Copper Domestic (a) Refinery	Electrolytic Copper Export (b) Refinery	Straits Tin New York	Lead New York	Lead St. Louis	Zinc St. Louis
1	11.775	11.200	52.000	5.85	5.70	8.25
3	11.775	11.200	52.000	5.85	5.70	8.25
4	Holiday	11.200		Holiday		
5	11.775	11.200	52.000	5.85	5.70	8.25
6	11.775	11.200	52.000	5.85	5.70	8.25
7	11.775	11.200	52.000	5.85	5.70	8.25
8	11.775	11.200	52.000	5.85	5.70	8.25
10	11.775	11.200	52.000	5.85	5.70	8.25
11	Holiday	11.200		Holiday		
12	11.775	11.200	52.000	5.85	5.70	8.25
13	11.775	11.200	52.000	5.85	5.70	8.25
14	11.775	11.200	52.000	5.85	5.70	8.25
15	11.775	11.200	52.000	5.85	5.70	8.25
17	11.775	11.200	52.000	5.85	5.70	8.25
18	11.775	11.200	52.000	5.85	5.70	8.25
19	11.775	11.200	52.000	5.85	5.70	8.25
20	Holiday	11.200		Holiday		
21	11.775	11.200	52.000	5.85	5.70	8.25
22	11.775	11.200	52.000	5.85	5.70	8.25
24	11.775	11.200	52.000	5.85	5.70	8.25
25	11.775	11.200	52.000	5.85	5.70	8.25
26	11.775	11.200	52.000	5.85	5.70	8.25
27	11.775	11.200	52.000	5.85	5.70	8.25
28	11.775	11.200	52.000	5.85	5.70	8.25
29	11.775	11.200	52.000	5.85	5.70	8.25
AVERAGES FOR MONTH						
Nov.	11.775	11.200	52.000	5.850	5.700	8.250
AVERAGES FOR WEEK						
Nov.						
5	11.775	11.200	52.000	5.850	5.700	8.250
12	11.775	11.200	52.000	5.850	5.700	8.250
19	11.775	11.200	52.000	5.850	5.700	8.250
26	11.775	11.200	52.000	5.850	5.700	8.250
CALENDAR WEEK AVERAGES						
Nov.						
1	11.775	11.200	52.000	5.850	5.700	8.250
8	11.775	11.200	52.000	5.850	5.700	8.250
15	11.775	11.200	52.000	5.850	5.700	8.250
22	11.775	11.200	52.000	5.850	5.700	8.250
29	11.775	11.200	52.000	5.850	5.700	8.250

SILVER, GOLD AND STERLING EXCHANGE

1941 Nov.	Sterling Exchange "Checks" (Nominal)	Sterling Exchange "90-day demand"	Silver (c) New York	Silver (c) London	Gold London	Gold (d) United States
1	401.000	(f)	(e)	(e)	(e)	$35.00
3	401.000	(f)	34.750	23.500	168s	35.00
4	Holiday	(f)	Holiday	23.500	168s	Holiday
5	401.000	(f)	34.750	23.500	168s	35.00
6	401.000	(f)	34.750	23.500	168s	35.00
7	401.000	(f)	34.750	23.500	168s	35.00
8	401.000	(f)	(e)	(e)	(e)	35.00
10	401.000	(f)	34.750	23.500	168s	35.00
11	Holiday	(f)	Holiday	23.500	168s	Holiday
12	401.000	(f)	34.750	23.500	168s	35.00
13	401.000	(f)	34.750	23.500	168s	35.00
14	401.000	(f)	34.750	23.500	168s	35.00
15	401.000	(f)	(e)	(e)	(e)	35.00
17	401.000	(f)	34.750	23.500	168s	35.00
18	401.000	(f)	34.750	23.500	168s	35.00
19	401.000	(f)	34.750	23.500	168s	35.00
20	Holiday	(f)	Holiday	23.500	168s	Holiday
21	401.000	(f)	34.750	23.500	168s	35.00
22	401.000	(f)	(e)	(e)	(e)	35.00
24	401.000	(f)	34.750	23.500	168s	35.00
25	401.000	(f)	34.750	23.500	168s	35.00
26	401.000	(f)	34.750	23.500	168s	35.00
27	401.000	(f)	34.750	23.500	168s	35.00
28	401.000	(f)	35.125	23.500	168s	35.00
29	401.000	(f)	(e)	(e)	(e)	35.00
AVERAGES FOR MONTH						
Nov.	401.000	....	34.772	23.500	....	35.00
AVERAGES FOR WEEK						
Nov.						
5	401.000	....	34.750	.....	....	.....
12	401.000	....	34.750	.....	....	.....
19	401.000	....	34.750	.....	....	.....
26	401.000	....	34.750	.....	....	.....

Calendar week averages, New York Silver: Nov. 1st, 34.750; 8th, 34.750; 15th, 34.750; 22nd, 34.750; 29th, 34.825.

(e) Not quoted (Saturday). (f) No quotations.

THE above quotations for major non-ferrous metals are our appraisal of the important United States markets, based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices are in cents per pound.

(a) Net prices at refineries on Atlantic seaboard. To arrive at the delivered New England basis add 0.225c. per pound, the average differential for freight and delivery charges.

(b) Export prices are net at refineries on the Atlantic seaboard and include sales of domestic copper in the foreign market. Owing to the European War and the disruption of normal trade relations, our export copper quotations, since September, 1939, have been based largely on f.a.s. transactions, ex United States ports. To arrive at the f.o.b. refinery quotations deduct .05c. from the f.a.s. basis for lighterage.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for copper are for the ordinary forms of wirebars and ingot bars; cathodes are sold at a discount of 0.125c.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York commands a premium over the St. Louis basis equal to the freight differential. Contract prices for High Grade Zinc delivered in the East and Middle West in nearly all instances command a premium of 1c per pound over the current market for Prime Western but not less than 1c over the *Engineering and Mining Journal's* average quotation for Prime Western for the previous month.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

(c) Silver other than newly-mined domestic, by Handy & Harman. Under Treasury order of July 6, 1939, the price on domestic newly-mined silver mined subsequent to July 1, 1939, was fixed at 71.11c per troy ounce. Handy & Harman's quotation on newly-mined domestic silver, 999 fine was 70⅝c throughout November.

(d) U. S. Treasury's gold price. Actual payment by the United States Treasury for gold in domestic and imported ore or concentrate is at 99.75 per cent of the price quoted by the Treasury, which at present is equal to $34.9125.

LONDON MARKET

Nov. 1941	Tin—Standard—Spot	Tin—Standard—3 months
3	255.7500	259.2500
4	256.0000	259.7500
5	256.0000	259.7500
6	255.7500	259.5000
7	255.5000	259.5000
10	256.0000	259.5000
11	256.5000	260.0000
12	257.2500	260.5000
13	258.5000	261.5000
14	258.0000	261.0000
17	257.2500	260.2500
18	256.7500	259.7500
19	257.2500	260.2500
20	257.2500	260.7500
21	257.7500	260.7500
24	257.0000	261.0000
25	256.5000	260.7500
26	256.2500	260.5000
27	256.5000	260.7500
28	256.5000	260.0000
Average for month	256.713	.....

Official tin prices on London Metal Exchange in long tons, pound sterling. Trading in copper, lead, and zinc suspended August 31, 1939.

Weekly MARKET INFORMATION Service

- E&MJ Metal & Mineral Markets, the weekly price information service of Engineering & Mining Journal, will keep you in close touch with effects of war developments on costs and availability of metals and non-metallic minerals.
- Printed and mailed every Wednesday night, E&MJ Metal & Mineral Markets will speed to you as special newspaper mail. It carries daily and weekly average prices of metals for the preceding six market days.
- Rates: $5 a year in the United States and possessions, and in the Americas; $10 a year elsewhere. Address:

E&MJ METAL & MINERAL MARKETS

Dept. 101 330 West 42nd Street New York, N. Y.

CURRENT PRICES — MISCELLANEOUS METALS, ORES, AND NON-METALLIC MINERALS

Quotations cover wholesale lots, prompt shipment, f.o.b. New York, unless otherwise stated

(Dec. 1, 1941)

MISCELLANEOUS METALS

Aluminum, ingot, 99 plus per cent, lb	15c.
Antimony, domestic, spot, lb	14.00c.
Bismuth, ton lots, lb	$1.25
Cadmium, commercial sticks, lb	90c.
Calcium, lb., ton lots 97 @ 98 per cent	$1.25
Chromium, 97 per cent grade, lb	80c.
Cobalt, 97 to 99 per cent, per lb	$1.50
Nickel, electrolytic cathodes, lb	35c.
Magnesium, 99.8 per cent, carloads, lb	27c.
Palladium, troy oz	$24.00
Platinum, (Official quotation) troy oz	$36.00
Quicksilver, flask of 76 lb., 100 flasks or more	$197.00
Radium, mg. radium content	$25.00@$30.00
Selenium, 99.5 per cent, lb	$1.75
Silicon, minimum 97 per cent, spot, lb	16.50c.
Tellurium, lb	$1.75
Thallium, 100 lb. or more, lb	$10.00
Titanium, 96 to 98 per cent, lb	$5.00

METALLIC ORES

Beryllium Ore, f.o.b. mines, ton	$30.00@$35.00
Chrome Ore, 48 @ 50% Cr_2O_3 c.i.f. Atl. ports, long ton	(b)$45.00@$47.00
Iron Ore, Lake Superior, Lower Lake ports, long ton:	
Old Range bessemer	$4.75
Mesabi bessemer	$4.60
Old Range non-bessemer	$4.60
Mesabi, non-bessemer	$4.45
Lead (Galena) 80 per cent, Joplin, Mo., ton	$68.64
Manganese Ore, (foreign) c.i.f. U.S. ports, long ton unit of Mn:	
52 @ 55 per cent	(b) 68c.
50 @ 52 per cent	(b) 68c.
46 @ 48 per cent	(b) 63c.
Molybdenum Ore, 90%, per lb. of MoS_2, f.o.b. mines	45c.
Tungsten Ore, per unit of WO_3:	
Chinese, 65 per cent, duty paid	(b) $24.00
Domestic, 65 per cent and upward	(a)$26.00@$27.00
Vanadium Ore, per lb. of contained V_2O_5	27½c.
Zinc Ore, Prime, 60 per cent concentrate, Joplin, Mo.; per ton	$55.28

(a) Prices at mines, small lots, usually several dollars less. (b) Nominal.

METALLIC COMPOUNDS

Arsenious Oxide (arsenic) lb	4c.
Cobalt Oxide, 70 @ 71 per cent, lb	$1.84
Copper Sulphate, 100 lb	$5.15
Sodium Nitrate, ex vessel, in 200-lb. bags, per 100 lb	$1.43
Sodium Sulphate, bulk, ton	$18.00@$24.00

ALLOYS

Beryllium Copper, master alloy, 2.5 to 3 per cent Be, per lb. of contained Be	$15.00
Ferrochrome, 65 @ 70 per cent chromium, 4 @ 6 per cent carbon, per pound of Cr contained	13c.
Ferromanganese, 78 @ 82 per cent, gross ton	$120.00
Ferromolybdenum 55 @ 65 per cent Mo. lb. of Mo. contained	95c.
Ferrosilicon, 50 percent, gross ton	$74.50
Ferrotungsten, 75 @ 80 per cent, lb. of W contained	$1.90
Ferrovanadium, per lb. of V, delivered	$2.70@$2.90

NON-METALLIC MINERALS

Asbestos, f.o.b. Canadian (Quebec) mines (U.S. funds), ton;	
Crude No. 1	$700@$750
Crude No. 2	$150@$350
Spinning fibers	$110@$200
Paper stock	$40@$49.50
Shorts	$13@$17.50
Vermont, f.o.b. Hyde Park:	
Shingle stock	$57.00@$60.00
Paper stock	$40.00@$48.00
Shorts	$13.00@$26.00
Floats	$18.00
Barytes, long ton:	
Georgia, crude	$8.00
Missouri, 93 per cent $BaSO_4$, less than 1 per cent iron	$6.00@$6.50
Bauxite, long ton:	
Domestic, crude, 50 @ 52 percent (not dried)	$5.00
Domestic, chemical, 55 @ 58 per cent	$7.50@$8.50
Domestic, abrasive, 80 @ 84 per cent	$16.00
China Clay, f.o.b. mines, ton:	
South Carolina and Georgia, No. 1, bulk	$6.75@$8.00
Delaware, No. 1	$14.50
Feldspar, bulk, ton:	
Potash feldspar, 200 mesh	$17.00
Glass-spar, white, 20 mesh	$11.75
Fluorspar, f.o.b. mines, bulk, Kentucky and Illinois 85–5 per cent, all rail movement, ton	$23.00
Ground, 95 @ 98 percent, bulk, ton	$32.00
Lump, acid, 98 percent, bulk, ton	$30.00
Fuller's earth, f.o.b. Georgia or Florida, ton	$7.00@$14.00
Magnesite, per ton:	
Dead-burned, f.o.b. California	$25.00
Dead-burned, f.o.b. Washington	$22.00
Mica, per lb., North Carolina, No. 1 and 2 quality:	
1½ x 2 in.	45@65c.
2 x 2 in.	60@85c.
3 x 3 in.	$1.85@$2.05
3 x 4 in.	$2.15@$2.25
3 x 5 in.	$2.50@$2.75
White, ground, 70 mesh, ton	$60.00@$80.00
Ocher, Georgia, ton	$19.00@$22.00
Pyrites, Spanish, per long ton unit of S, c.i.f. Atlantic ports	(a) 12c
Silica, in bags, 325 mesh, ton	$20.00@$40.00
Sulphur, Texas, mines, long ton	$16.00
Talc, f.o.b. works, ton:	
New York, double air-floated, 325 mesh	$12.00@$15.00
New Jersey, mineral pulp	$8.50@$10.50
Vermont, extra white, 200 mesh	$9.50@$10.00
Tripoli, Missouri, ton:	
40 mesh, cream colored	$14.50
200 mesh, cream colored	$26.00

(a) Nominal

IRON AND STEEL

Pig Iron, Valley furnaces, gross ton:	
Bessemer	$24.50
Basic	$23.50
No. 2 Foundry	$24.00
Steel, base prices, Pittsburgh:	
Billets, gross ton	$34.00
Structural shapes, 100 lb	$2.10
Bars, 100 lb	$2.15

Monthly and Yearly

AVERAGE PRICES

SILVER AND STERLING EXCHANGE

	New York		London Spot		Sterling Exchange	
	1940	1941	1940	1941	1940	1941
January	34.750	34.750	21.892	23.273	395.442	(a) 402.000
February	34.750	34.750	20.935	23.341	395.652	(a) 401.864
March	34.750	34.750	20.763	23.446	375.212	(a) 401.788
April	34.750	34.750	20.713	23.500	351.817	(a) 401.058
May	34.949	34.750	21.878	23.457	326.452	(a) 400.981
June	34.825	34.750	22.688	23.400	359.560	(a) 401.000
July	34.750	34.750	22.095	23.397	379.750	(a) 401.000
August	34.750	34.750	23.261	23.459	396.889	(a) 401.000
September	34.750	34.750	23.446	23.500	401.646	(a) 401.000
October	34.750	34.750	23.451	23.500	401.692	(a) 401.000
November	34.750	34.772	23.238	23.500	401.783	(a) 401.000
December	34.750	.....	23.015	.....	402.000	.....
Year	34.773	.....	22.281	.....	382.325	.....

New York quotations for silver other than newly-mined domestic, cents per ounce troy, 999 fine. London pence per ounce sterling silver 925 fine. Sterling exchange (pound sterling) in cents. (a) Nominal

COPPER

	F.O.B. Refinery, Electrolytic			
	Domestic		Export	
	1940	1941	1940	1941
January	11.954	11.819	11.999	10.257
February	11.148	11.794	11.471	10.414
March	11.160	11.814	11.407	10.592
April	11.087	11.820	11.258	10.952
May	11.079	11.815	11.191	10.950
June	11.128	11.810	11.216	10.950
July	10.564	11.812	10.189	10.950
August	10.708	11.778	9.851	10.950
September	11.296	11.775	9.849	11.027
October	11.826	11.775	10.436	11.367
November	11.800	11.775	10.084	1.200
December	11.802	.....	10.293	.....
Year	11.296	.....	10.770	.....

New York quotations, cents per pound.

LEAD

	New York		St. Louis	
	1940	1941	1940	1941
January	5.471	5.500	5.321	5.350
February	5.076	5.602	4.926	5.452
March	5.192	5.765	5.042	5.613
April	5.071	5.850	4.921	5.700
May	5.015	5.850	4.865	5.700
June	5.000	5.850	4.850	5.700
July	5.000	5.850	4.850	5.700
August	4.854	5.850	4.704	5.700
September	4.929	5.850	4.779	5.700
October	5.308	5.850	5.158	5.700
November	5.726	5.850	5.578	5.700
December	5.500	.....	5.350	.....
Year	5.179	.....	5.029	.....

New York and St. Louis quotations, cents per pound

TIN

	Straits New York		Standard, Spot London	
	1940	1941	1940	1941
January	46.707	50.154	240.716	256.648
February	45.851	51.293	242.833	264.975
March	47.079	52.067	251.711	270.131
April	46.815	51.981	252.080	269.775
May	51.570	52.166	264.098	267.534
June	54.618	52.685	273.438	262.750
July	51.591	53.481	265.592	258.272
August	51.176	52.385	262.455	257.013
September	50.348	52.000	251.024	256.364
October	51.490	52.000	257.946	255.842
November	50.577	52.000	258.214	256.713
December	50.101	.....	256.988	.....
Year	49.827	.....	256.425	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

ZINC

	St. Louis		
	1939	1940	1941
January	4.500	5.644	7.250
February	4.500	5.534	7.250
March	4.500	5.750	7.250
April	4.500	5.750	7.250
May	4.500	5.803	7.250
June	4.500	6.235	7.250
July	4.516	6.250	7.250
August	4.719	6.389	7.250
September	6.104	6.920	7.250
October	6.500	7.250	7.942
November	6.500	7.250	8.250
December	5.980	7.250	.....
Year	5.110	6.335	.....

St. Louis quotations, Prime Western, cents per pound.

CADMIUM AND ALUMINUM

	Cadmium		Aluminum	
	1940 (a)	1941 (a)	1940	1941
January	79.038	80.000	20.000	17.000
February	80.000	82.273	20.000	17.000
March	80.000	89.038	19.769	17.000
April	80.000	90.000	19.000	17.000
May	80.000	90.000	19.000	17.000
June	80.000	90.000	19.000	17.000
July	80.000	90.000	19.000	17.000
August	80.000	90.000	18.000	17.000
September	80.000	90.000	18.000	17.000
October	80.000	90.000	18.000	15.000
November	80.000	90.000	17.522	15.000
December	80.000	.....	17.000	.....
Year	79.920	.....	18.691	.....

Aluminum in cents per pound, 99 plus per cent grade. Cadmium, cents per pound. (a) Producers' price, commercial sticks.

ANTIMONY, QUICKSILVER, AND PLATINUM

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1940	1941	1940	1941	1940	1941
January	14.000	14.000	156.962	165.846	40.000	36.000
February	14.000	14.000	178.000	170.182	40.000	36.000
March	14.000	14.000	180.921	177.692	40.000	36.000
April	14.000	14.000	173.538	180.077	38.923	36.000
May	14.000	14.000	181.538	180.000	38.000	36.000
June	14.000	14.000	197.360	183.920	38.000	36.000
July	14.000	14.000	194.423	188.577	38.000	36.000
August	14.000	14.000	184.111	192.000	38.000	36.000
September	14.000	14.000	173.333	192.440	36.167	36.000
October	14.000	14.000	168.846	193.615	36.000	36.000
November	14.000	14.000	168.391	196.273	36.000	36.000
December	14.000	.....	164.960	.....	36.000	.....
Year	14.000	.....	176.865	.....	37.924	.....

(a) Antimony, cents per pound, ordinary brands, in cases; in bulk f.o.b Laredo, Tex., 13.000 for November. (b) Quicksilver, dollars per flask of 76 lb. (c) Platinum, per ounce troy.

PIG IRON

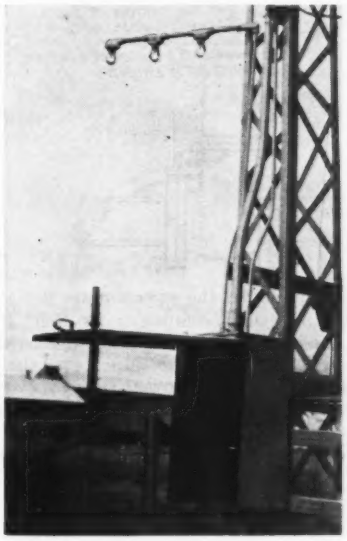
	Bessemer		Basic		No. 2 Foundry	
	1940	1941	1940	1941	1940	1941
January	23.50	24.50	22.50	23.50	23.00	24.00
February	23.50	24.50	22.50	23.50	23.00	24.00
March	23.50	24.50	22.50	23.50	23.00	24.00
April	23.50	24.50	22.50	23.50	23.00	24.00
May	23.50	24.50	22.50	23.50	23.00	24.00
June	23.50	24.50	22.50	23.50	23.00	24.00
July	23.50	24.50	22.50	23.50	23.00	24.00
August	23.50	24.50	22.50	23.50	23.00	24.00
September	23.50	24.50	22.50	23.50	23.00	24.00
October	23.50	24.50	22.50	23.50	23.00	24.00
November	23.50	24.50	22.50	23.50	23.00	24.00
December	23.50	.....	22.50	.....	23.00	.....
Year	23.500	.....	22.500	.....	23.000	.....

Iron, dollars per long ton. F.o.b. Mahoning and Chenango Valley furnaces

USEFUL OPERATING IDEAS

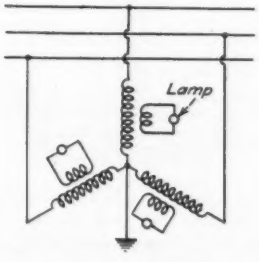
Lamps Signal Grounds At Phosphate Mine

SAFETY TO MEN, low cost of maintaining equipment, and avoidance of delays call for immediate detection and clearing of a ground on a 2,300-v. distribution system to motors and transformers. The accompanying diagram shows the connections of potential transformers and



Door opened to show potential transformers mounted below the indicating lamps at Ridgewood substation

ground-detector lamps used at each of the 33,000- to 2,300-v. transformer substations of the Southern Phosphate Corp., Bartow, Fla. Secondaries of these power transformers are connected delta, as is the common practice. The photograph shows the potential transformer protective box and the lamp mounting at the Ridgewood drying plant of that corporation.



Connections of potential transformers and lamps

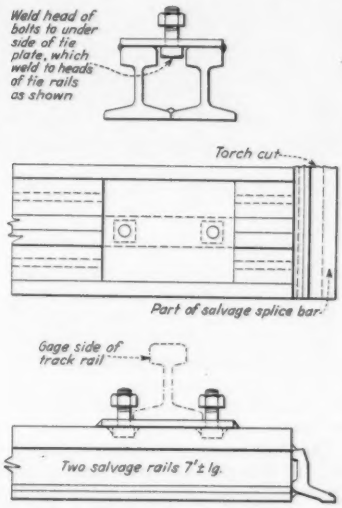
Assuming that the line voltage is 2,300 and the system is free of a ground, each lamp is lighted at 64 v. But if a conductor becomes grounded the lamp powered by the potential transformer connected to that conductor dims or goes out, depending on resistance of the accidental ground. If this ground is a "bad" one, the other two potential transformers get full voltage and their lamps brighten by getting 110 v.

▼

Railroad Track Tie Made of Salvage Rail

UNDER loading and waste dump tracks which are being constantly shifted a timber tie wears out rapidly, says Elwood Stockman. The accompanying sketch shows a welded steel track tie developed by the Mines Department of the Chile Exploration Co. at the Chuquicamata open-pit mine.

As shown it is made entirely of steel, and all joints are welded. The


Weld head of bolts to under side of tie plate, which weld to heads of tie rails as shown
Torch cut
Part of salvage splice bar
Gage side of track rail
Two salvage rails 7' ± lg.

Illustrating track-tie construction

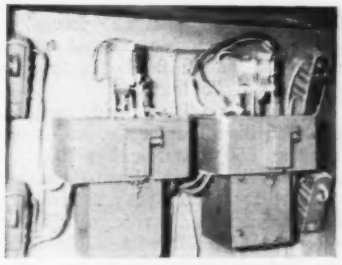
backbone consists of two salvage rails spot-welded at their abutting edges, welded together at their ends by two suitable lengths of used-track splice bar and welded together across their heads by two used tie plates. The tie plates are drilled for suitable bolts, the heads of which are spot-welded before the plates are welded to the rails. Bolt centers are made to suit weight of track rails and style of rail clips.

In addition to its capacity for taking a lot of punishment, this steel tie has the added advantage of a base fully as wide as a timber tie for bearing on the ballast.

▼

Solenoids Improve Operating Switches

CHIEF ELECTRICIANS get gray hairs trying to educate equipment operators to open and close electric switches with a quick decisive throw instead of with a slow or teasing motion which burns contacts. Safer operation, remote control where desired and elimination of burned contacts of 2,300-v. FK20 oil switches



Manual switches converted to pushbutton operation

were achieved at a cost approximating $25 per switch at San Gully mine and plant of the Southern Phosphate Corp. near Lakeland, Fla.

In the illustration showing two of three switches which have thus far been fitted at San Gully, the solenoids are in evidence on the tops of the covers. These a.c. solenoids, designated as CR-9503-206A and purchased from General Electric Co., have the following specifications: 22.7-lb. pull at 85 percent voltage; 30-lb. pull at 100 percent voltage; 1¼-in. stroke; shipping weight 10 lb.

These two switches are mounted on the wall inside of a steel switch room on the washer. One controls a 75-hp. motor driving the mill and another a 25-hp. pump motor, both operated at 2,300 v.

Fuel Saving $5 Daily With New Diesel

INSTALLATION of a high-speed diesel engine late last year to replace a gasoline engine as the power source for a tailings mill is saving $5 per day in fuel cost, according to William Butler, Jr., who in partnership with F. B. Moody, Jr., operates the "Little Mill" at Mulliken, Ky. This new unit, rated 60 hp., replaced a 40-hp. engine which was too light for the duty. The mill recovers 5 to 6 tons of flux-grade fluorspar (85 percent or more calcium fluoride and not over 5 percent silica) in 24 hours from a total feed of 75 tons.

"Little Mill" with Klondyke mine and mill at upper right. In the left foreground is the pile of reworked tailings

The diesel is a Type-D8800 made by the Caterpillar Tractor Co. and is operated at 650 r.p.m. with a flat belt drive to a line shaft. A small generator belted from the front end furnishes power for lighting. Principal items of equipment driven by the engine are: one Tyler-Niagara vibrator, three Harz-type jigs, one Norman-type jig, one diaphragm jig, one Denver pulsating mineral jig, hydraulic classifier, one Deister concentrator table, crushing rolls, and a belt conveyor handling refuse to the dump.

By means of a gasoline-engine-driven portable loading conveyor, material from the old tailings pile is loaded into a motor truck and dumped into an elevated feed hopper. From there it is sluiced to the elevator delivering to the vibrator, which makes four sizes. These then move as follows: ½x¼ in. to No. 1 Harz roughing jig, ¼x⅛ in. to No. 2 Harz roughing jig, ⅛x⅛-in. to Norman jig, ⅛x0-in. to a hydraulic classifier which takes out a 32x0-in. to be concentrated on the table.

The diesel furnishes more power to the mill at considerably lower fuel cost

Power for lighting is furnished by a 2-hp. d.c. motor, also driven by the engine

The ⅛x32 goes to the diaphragm jig and its concentrate then gets a final treatment in the Denver pulsating jig. Concentrate from No. 1 Harz jig is crushed to ¼ in. and returned to the vibrator feed. Concentrate from the Harz ¼x⅛ in. roughing jig is finished in the third Harz jig.

The old tailings pile is that from Klondyke mine, an old operation shut down during the past year and at one time the largest fluorspar producer in Kentucky. It is not on a railroad but is only one mile from a large terminal on the Cumberland River. The locality is about 5 miles southwest of Salem. Mr. Butler was master mechanic for the Klondyke Co.

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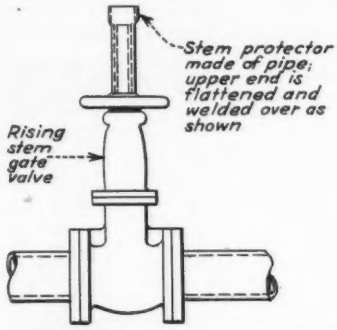
Protecting Threads on Gate Valves

TO PROTECT the threads on semi-permanently opened rising-stem gate valves in very dusty locations around its reduction works, writes Elwood Stockman, the Chile Exploration Co., Chuquicamata, Chile,



has developed the very simple device shown in the accompanying sketch. It consists of a pipe of suitable diameter and length, with the upper end flattened and welded over, slipped over the protruding stem. Thus, the forming of the harmful dust-grease combination is prevented and the life of the stem and its nut is prolonged.

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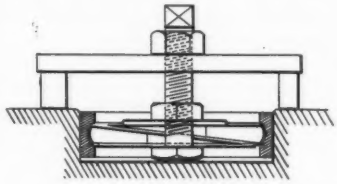
Broken Ball Bearings Removed From Housings

BALL-BEARING FAILURE usually leaves the inner race on the shaft and the outer race in its housing. As there was no means of pressing the race out from behind,



a special withdrawing device was made to accomplish the job at the Great Boulder Gold Mines, Australia, writes Roger J. Burton.

The device consists of two slotted

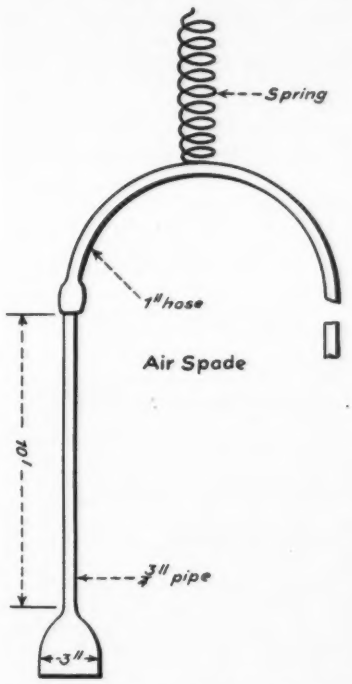
wedge pieces, shaped at their outer ends to fit the curvature of the ball path in the outer race.

The procedure is to locate the wedges in the race, as per cut, where they are locked into position by the bolt, nut, and covering washers. The bolt is made with a square end and of sufficient length to accommodate a draw plate, whereby the entire assembly is then extracted from the housing.

▼

A Handy Air Spade

IN MANY MINE OPERATIONS, writes Paul C. Ziemke, Milwaukee, Wis., the material handled is somewhat plastic in nature and as a result tends to cling to the walls and doors of storage raises, chutes, slides, and



aprons. The cleaning of these can be expedited greatly by the use of the air spade, which consists of a ¾-in. pipe about 10 ft. long, one end of which is flattened by the blacksmith to form a spade or cutting edge while the other is clamped into a length of 1-in. hose.

Some workmen like a valve at fingertip reach; others prefer it at point of attachment of hose to supply pipe. For those places where it is in almost constant use the spade is suspended by means of either a spring or a counter-balancing weight.

In service the force of the compressed air getting under the accumulated coatings of material blows this off in large sheets and thereby vastly speeds up the work of "cleaning down."

Spiral Classifier in Dewatering Duty Raises Quality of Concentrates

Dewaters the concentrates and lowers the silica content

DEWATERING with a 30-in. Akins classifier as is now the practice at a mill of the Crystal Fluorspar Co., Cave-In Rock, Ill., is a new use for that type of unit in fluorspar mining. An incidental advantage is that some of the silica is removed by this dewatering method.

Formerly, fluxing grade fluorspar concentrates were laundered to a stock-

The classifier is in this annex and it discharges to the belt conveyor boom

pile in the yard. Disadvantages were a low height limit to the stockpile and the fact that the water draining from the pile created a sloppy nuisance in the mine yard.

An elevated annex was built to the mill to house the classifier and a cantilever supported belt conveyor installed to take the concentrates from the classifier discharges to the stockpile.

Material of minus 150 mesh leaves with the water and these solids show a 20 percent silica content. The concentrates turned out by the mill are of a grade averaging 4 percent silica.

Output of this mill of the Crystal Fluorspar Co. is 60 tons per 8-hour shift. The ore is mined in three shafts all within ⅜ mi. of the mill. Mining and milling operations are under the supervision of D. G. Gibson, Jr., and Miles Haman. M. M. McClelland is mining engineer.

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Air Valve Protector

THE AIR VALVES on electric shovel-loaded ore cars of the Chile Exploration Co., Chuquicamata, Chile, were constantly being

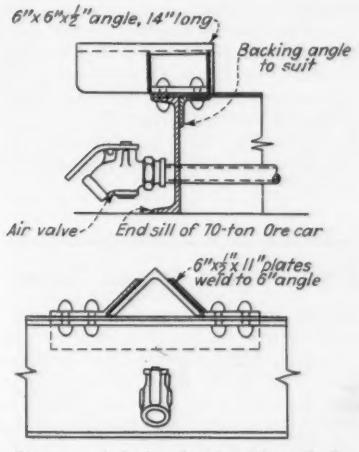


Diagram of device for protection of air valves

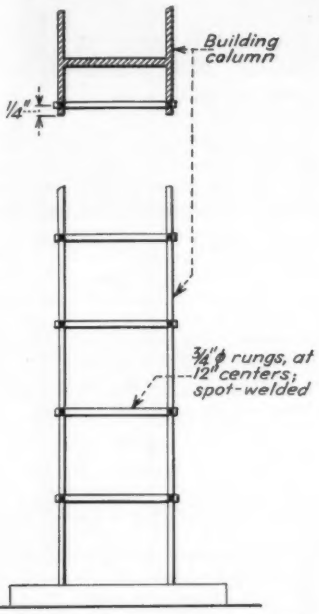
damaged by large-size rock falling on them during loading operations. Many costly delays resulted, states Elwood Stockman.

To eliminate this source of trouble an inverted V-type protector made of ½ in. steel was riveted to the end sill of the 70-ton ore cars, as shown in the accompanying sketch. This simple device has proved to be satisfactory, as all falling material is now deflected from the air valve.

▼

Steel Ladder

THE ever-present problem of the borrowed step-ladder, which is never returned, has been overcome to some extent by resorting to the device shown in the accompanying sketch, submitted by Elwood Stockman. Also, it serves very well in necessarily crowded locations. The rungs should be slightly longer than the depth of the column, to allow for spot-welding. It is in regular use at the original secondary crushing plant of the Chile Exploration Co., Chuquicamata, Chile. It is recommended for use on columns of twelve inches depth and over.

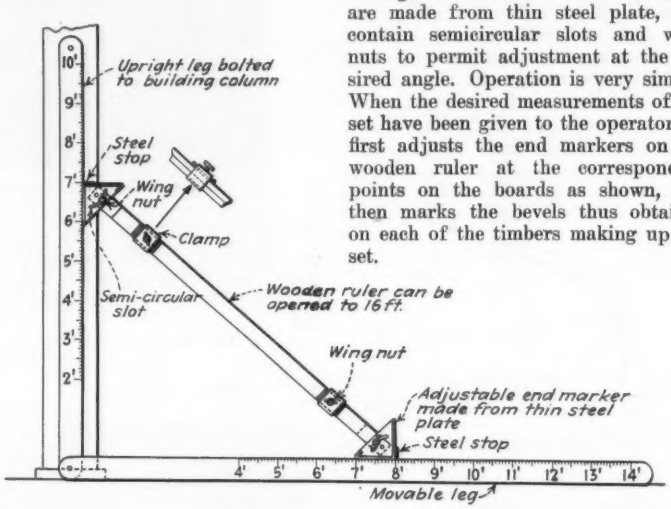


Five Dollars for an "Idea"

SEND your contribution. *E.&M.J.* pays a five dollar minimum for all operating ideas accepted for publication. Just sketch it, or photograph it, and describe it in sufficient detail to give the complete idea, then send it in. Our readers want it.

A Device for Marking Bevels of Segment Sets

A SIMPLE and dependable arrangement that has speeded up materially preparation of segment sets used in stopes, drifts, and tunnels was recently developed at the timber framing shop of the United Verde Branch, Phelps Dodge Corp., Jerome, Ariz. Confronted with the task of making up periodically a large number of segment sets of different size and design, and knowing the time and labor required to lay out the correct bevels on each timber, Mike Amersek, an employee, evolved the ingenious device illustrated in a self-explanatory manner in the accompanying sketch. It essentially consists of two graduated, planed narrow boards, one bolted in an upright position to one of the columns supporting the roof of the shop building and the other movable and in a horizontal position when the device is in operation, and the marking unit proper. All graduation marks (feet and inches) on the boards are notched in with a thin saw to accommodate the steel stops holding in place at designated points the markers at the ends of the adjustable wooden ruler. This ruler is fitted with two stop clamps, and can be opened to a length of 16 ft. The end markers are made from thin steel plate, and contain semicircular slots and wing nuts to permit adjustment at the desired angle. Operation is very simple. When the desired measurements of the set have been given to the operator, he first adjusts the end markers on the wooden ruler at the corresponding points on the boards as shown, and then marks the bevels thus obtained on each of the timbers making up the set.

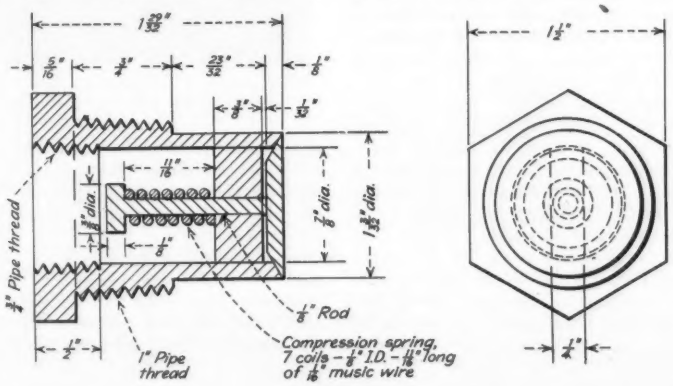


Check Valve for Greasing Lower Tumblers on Dredges

TO MAINTAIN PRESSURE after greasing the lower tumblers on a gold dredge the attachment shown in cut was worked out and applied by Frank B. Dibble, of LaGrange, Stanislaus County, Calif.

An ordinary pipe bushing of any size may be used, to which guide, rod, valve, and spring are added by brazing. In this instance a 1x¾ in. pipe bushing was used with a seven-coil compression spring to give the general assembly.

A marked saving in the amount of grease applied to the lower tumblers was proved at this operation.



PERSONAL ITEMS

The British Institution of Mining and Metallurgy has established temporary offices at the Royal School of Mines, London S.W. 7, England.

Howard Mellor has been elected vice president of the Illinois Zinc Co.

Ernest Oberbillig, author of the article "Recovering Slime Concentrate by Flotation," was in New York recently.

George F. Reed is rehabilitating and developing the Lion's Den mercury mine, near Solvang, Calif.

W. R. Wright is now in the Canal Zone, after having examined some properties in Panama.

William J. Loring has become managing mining engineer for Madre de Oro Mines, Inc., at Eldorado, Calif.

R. O. Brandon, general foreman at the Hill-Annex mine, has been appointed assistant superintendent under **E. S. Tillinghast.**

C. Bruder has changed his address from Winnipeg, to Jason Mines, Ltd., Casummit Lake, Ont. His new location is 100 miles from railroad.

W. G. Stuart, in charge of operations for Mercury Gold Mines, Ltd., in the Northwest Territories, is spending the winter in Vancouver.

H. C. Enos, of Chatsworth, Calif., has just returned to Los Angeles after making mine examinations in the Bishop and Death Valley districts.

John Russell, of Springer Sturgeon Gold Mines, Ltd., has returned to Toronto, after prospecting in British Columbia during the past season.

Mahlon Miller, of San Diego, Calif., field engineer of the American Smelting & Refining Co., has recently been in British Columbia on a program of investigation for his company.

R. W. Michael, mining engineer in charge of the technical division of Cia. de Diamantes de Angola, recently returned to New York from Portuguese West Africa.

A. A. Hoffman has resigned as California manager of American Potash & Chemical Corp., Trona, Calif., and is now residing at 30 Bridge Road, Berkeley, Calif.

F. H. Stewart has left Pachuca, Mexico, where he has been geologist for Cia. Real del Monte, and has returned to Denver, Colo., where his address is 1215 Holly St.

Arnold H. Miller, consulting engineer of New York City, is in Mexico City, returning from Durango, where he has been on professional work for New York interests.

Dr. Elmer G. Peterson, president of Utah State Agricultural College, has been appointed a member of a national committee of ten to consider and recommend means of developing the phosphate resources of the Western States.

John B. Huttl, Assistant Editor of *Engineering and Mining Journal,* with headquarters in San Francisco, will make a tour of the mining districts of South America, attending also the first Pan American Congress of Mining Engineering and Geology in Santiago, Chile. Traveling by Pan American Airways, he will leave Miami, Florida, on December 16, and will visit Brazil, Argentina, Chile, Peru, Bolivia, Ecuador and Colombia.

JOHN B. HUTTL

Richard J. Anderson, instructor in geology at the University of Minnesota, has been appointed Assistant State Geologist for Arkansas, succeeding **Dr. George R. Heyl,** who has resigned to engage in commercial work. Mr. Anderson will take his new post Dec. 30.

Frank O. Horne, 73, superintendent of the United States mine, at Bingham, Utah, retired Nov. 1 after 22 years of service. **E. F. Kincaid,** superintendent of the U. S. Smelting, Refining & Mining Co.'s Hidden Treasure mine, at Ophir, succeeded Mr. Horne.

Edgar Y. Ellis, Joplin mining engineer, has been named superintendent of the Waco properties of the St. Louis Smelting & Refining Co. He succeeds **S. C. Sandusky,** of Baxter Springs, who will be in charge of exploration and geological work for the company.

Eugene McAuliffe, of Omaha, Neb., has been elected president of the American Institute of Mining and Metallurgical Engineers. Mr. McAuliffe is president of the Union Pacific Coal Co., the Washington Union Coal Co., and the Southern Wyoming Utilities Co.

John D. MacKenzie was recently made manager of the California department of the American Smelting & Refining Co. and president of Federated Metals Division of the company, with offices at 405 Montgomery St., San Francisco. He was until his promotion manager of the Southwestern department of the company, in which post he is succeeded by **R. D. Bradford,** recently smelter manager at East Helena, Mont.

Felix Williams, manager of the local plant of the Monsanto Chemical Co. at Columbia, Tenn., has been made manager of the entire phosphate department, with charge of plants at Anniston, Ala., St. Louis, Mo.; and Detroit, Mich., among others. **A. T. Beauregard** has become manager of the local plant at Monsanto, succeeding Mr. Williams.

Jesse Madson, superintendent of the Grant mine, at Buhl, Minn., has been transferred by the Inter-State Iron Co. to Star Lake, N. Y. He will be in charge of the Benson mines being developed there by the Inter-State Iron Co. **Paul Kruse,** mining engineer at the Hill-Annex mine, at Calumet, Minn., will succeed Mr. Madson as superintendent of the Grant mine.

William J. O'Connor, manager of the Utah department of the American Smelting & Refining Co., recently announced that **Edward M. Tittmann,** assistant manager, had been transferred to East Helena, Mont., to manage the company's smelter. **Kuno Doerr, Jr.,** assistant superintendent of the Garfield smelter, succeeded Mr. Tittmann. **R. A. Perry,** superintendent of the Murray smelter, was named to take Mr. Doerr's place at Garfield, and **F. J. Downey,** Garfield plant metallurgist, succeeded Perry.

Pierce C. Walker, mining and metallurgical engineer, will be transferred by the Standard Cyaniding Co., near Lovelock, Nev., as general superintendent for the Willow Creek Mining Co., which will operate the Consolidated Gold Acres mining property, 30 miles south of Beowawe, Nev. A 400-ton diesel-electric cyanide plant is being constructed at the property by Western Knapp Engineering Co. Milling is expected to start before the end of 1941.

Philip D. Wilson, vice president of Pardners Mines Corp., New York, has been appointed chief of production of the Aluminum and Magnesium Branch of OPM, responsible for the production of bauxite, alumina, aluminum, and magnesium, for all products fabricated therefrom, and for the supply of all ingredients necessary for their production. Mr. Wilson has been acting as chief engineer of the Bauxite and Alumina Unit since last July.

Roger W. Straus, vice chairman of the board of the American Smelting & Refining Co., was on Nov. 27 elected president of the company, succeeding the late Simon Guggenheim. He was also appointed chairman of the finance committee. On April 7, 1920, Mr. Straus was appointed assistant to the president and on Jan. 16, 1923 was made a member of the finance committee of the board of directors. On April 4, that year, he was elected a vice president and assistant to the president. In April, 1940, he was named vice chairman of the company.

H. L. Garrity was made superintendent of the Utah Copper open-cut mine

at Bingham, Utah, on Nov. 21. Mr. Garrity started with the company in 1924 as a draftsman. In 1934, he was made assistant superintendent of the Bingham & Garfield Railroad and in 1938 assistant superintendent of the Utah Copper mine. Other promotions made by **L. Buchman,** general superintendent, were **L. F. Pett,** to be general mine foreman; **V. S. Barlow,** general mill foreman, Utah section; and **J. C. Landenberger, Jr.,** general track foreman.

W. L. Batt, director of the division of Materials of OPM, has announced the following changes in personnel of the Division: **David A. Uebelacker,** present chief of the Nickel Branch, becomes chief of the Copper Branch, succeeding **John A. Church.** Mr. Church becomes senior consultant for the Copper Branch. **Louis Jordan,** assistant chief of the Nickel Branch, has been named acting chief. **Howard C. Sykes,** present chief of the Mica, Graphite Branch, moves over to become chief of the Stockpile and Shipping Imports Branch, succeeding **Philo W. Parker.** Mr. Parker has served his agreed term in OPM and is returning to private business. **Howard F. Wierum,** assistant chief under Mr. Sykes, replaces him as chief.

Ernest William Ellis has recently returned from Nicaragua, where he was manager of Coco Mines, Ltd. This property, a large-tonnage, low-grade, free-gold deposit, is some 8 miles back in the jungle from the native village of San Carlos, which is on the Rio Wanks, in the northeastern part of the country, some 230 miles up river. Transportation from the river to the mine and ore haulage are maintained with No. 70 Caterpillar tractors. Rainfall is heavy, reaching as high as 200 in. per year. Mr. Ellis contracted malaria and was taken by pit-pan (native dugout canoe) 100 miles down the Wanks River to the only hospital in that section, the Moravian Mission Hospital at Bilwas Karma. The trip was made during the worst hurricane that Nicaragua has had in years, even exceeding that of 1935. The date was Sept. 27 last. Prevalence of malaria necessitated Mr. Ellis' return to the United States. His address for the present is 714 East B St., Moscow, Idaho.

▼ ▼ ▼

OBITUARY

H. Connors, who was mill superintendent at Big Bell Mines, Ltd., Western Australia, was fatally injured in a mine accident on Sept. 16.

Major Edwin Clarence Eckel, chief geologist of the Tennessee Valley Authority, died on Nov. 22 at his farm near Knoxville, Tenn., at the age of 67.

Franklin Thompson Sutherland, a retired mining engineer active in the development of mining in the West in the early days of mining there, died at Jacksonville, Fla., on Nov. 18. He was 87.

James R. Peberdy was killed at Wahroonga, a suburb of Sydney, on Sept. 24. He had been for a number of years engaged in alluvial tin mining in Malaya and was latterly consultant to Tableland Tin Dredging N. L. in Queensland.

William A. Thomas, former mill superintendent at the Young-Davidson Mines, Ltd., in Matachewan, Ontario, Canada, died in Toronto General Hospital late in September after a long illness. He was born in Cornwall and came to Timmins, Ont., seventeen years ago, where he was employed by Hollinger Consolidated Gold Mines. He is survived by his wife.

John Alexander Lindsay, who was in the service of the Broken Hill Proprietary Company, Ltd., for a period of 43 years, died on July 27. Mr. Lindsay was for 33 years chief engineer to the company at its mine at Broken Hill and since 1934 had been consulting engineer to the company in Adelaide, supervising Whyalla construction contracts and mechanical orders.

Livingston Wernecke, of Berkeley, Calif., mining engineer, geologist, scientist, and mine operator, was killed in an airplane accident on Oct. 21 while flying from Hyder, Alaska, by way of Mayo, Yukon Territory. He was 58. He and his pilot, Charles Gropstis, crashed in a misty rain at Lady Douglas Island, in Milbank Sound, British Columbia. Mr. Wernecke was at the time of his death manager and consulting engineer for the Treadwell Yukon Corp. He had since 1907 been conspicuously identified with the development of mining in Alaska as well as with the construction of railroads there. He was associated also with Alaska Gastineau Mining Co., Alaska Treadwell Gold Mining Co., Alaska Juneau Gold Mining Co., all in Alaska, and with Beebe Gold Mining Co., Pacific Mining Co., and Atolia Mining Co., of which he was president, all operating in California. He was a member of the A.I.M.E. and the Canadian Institute of Mining and Metallurgical Engineers.

An Appreciation of Simon Guggenheim

Simon Guggenheim (born Dec. 30, 1867; died Nov. 2, 1941) was the sixth son of Meyer Guggenheim and one of that famous group of Guggenheim brothers whose names are so intimately associated with the development of many famous mining and metallurgical enterprises. In 1901, the then Guggenheim mining and smelting interests were consolidated with American Smelting & Refining Co. Senator Guggenheim at that time lived in Colorado and remained in charge of that most important division of the company until, in 1907, he was elected to the United States Senate. I have been told by important Colorado politicians of that era that Senator Guggenheim accomplished more concrete results for his State and constituents than any previous Colorado Senator. But the lost motion, delays, and inefficiency of government activities were the antithesis of his inborn characteristics, and so at the end of his term he declined to be a candidate for reelection, preferring to return to his business career. Accordingly, he went to New York and resumed his association with American Smelting & Refining Co., becoming chairman of the board of directors, his elder brother Daniel at that time being the president. Upon Daniel's retirement, Senator Guggenheim became president in January, 1919, and remained such until his death, a period of 23 years lacking about two months.

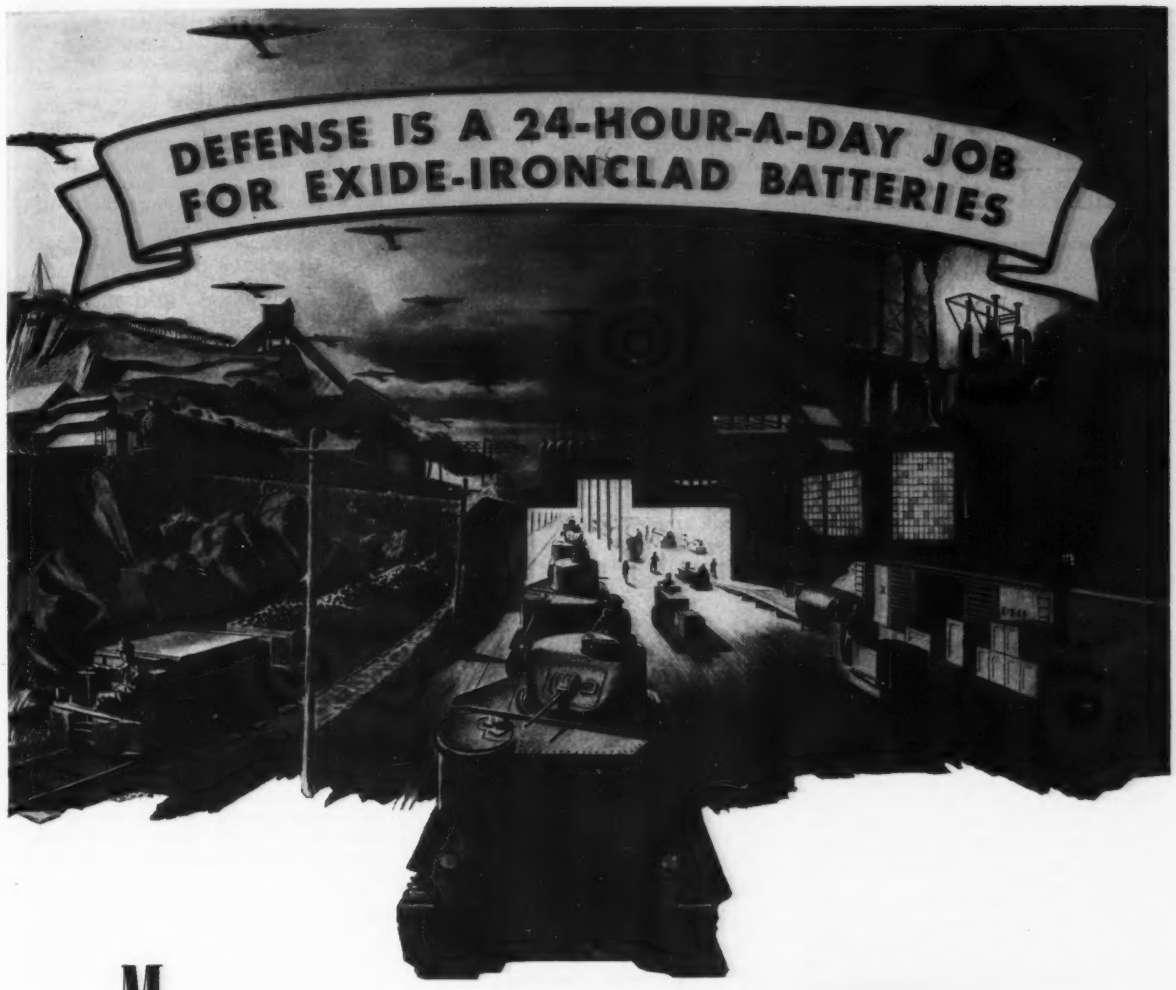
During his presidency, the Smelting company entirely paid off its bond issue, which at its maximum was over $52,000,000, built nine new plants of various kinds in the United States and Mexico, acquired large interests in General Cable Corporation and Revere Copper & Brass Incorporated, acquired the entire properties of Federated Metals Co., the largest secondary-metal business in the United States, and acquired many new mines or important interests therein in various parts of the world, including Newfoundland, Australia, Peru, Mexico, and Saudi Arabia, as well as the United States. The company paid out or declared in dividends during the Senator's presidency $178,276,028. It retained for uses of the company for its expansion and maintenance policy $194,556,712.

As an executive, Senator Guggenheim possessed marked sagacity in the choice of his leading lieutenants (to whom, once selected and proved, he left to an unusual degree the administration of their respective departments) and a marked ability to see, grasp, and appreciate fully the viewpoint of the other fellow, even though he may not have agreed with him. He always showed interest in the welfare of his subordinates, was always approachable and sympathetic in his dealings with them, and fair in his decisions where differences of opinion arose. Few leaders have left behind them a more loyal and devoted following.

The Senator was always generous in his philanthropies, in assistance to old friends, as well as in gifts to charitable and educational institutions. His name will go down to future generations as the founder, with Mrs. Guggenheim, of the John Simon Guggenheim Memorial Foundation, to which they gave some $8,000,000 during the Senator's lifetime, and to which Foundation, having no living children, he has left much the greater part of his fortune. The work of that Foundation in the advancement of knowledge and in giving opportunities to men and women to pursue special studies, after having demonstrated in actual life their intellectual ability, fills a field not occupied by any other Foundation. Already the great wisdom shown in giving for such a purpose is becoming apparent.

For over twenty years we had adjoining and connecting offices and were in almost daily consultation. No unpleasant incident ever arose between us. Not only in business, but also on the golf course and in the home, I had opportunity to know and appreciate his many fine qualities. He was a charming companion and his shrewd, penetrating comments upon men and affairs were a constant source of interest. As a true and loyal friend, he left nothing to be desired.

Simon Guggenheim's life was a full and well-rounded one. He was successful in business, served his nation in high political office, and gave millions of dollars for the benefit of mankind. He left behind a host of devoted and admiring friends, and his name will go on in the future associated not only with the successful administration of a large industrial company, but, of far



MORE TONS of coal, more tons of minerals . . . by the thousands . . . are needed. They must be mined and hauled *faster* to keep defense furnaces roaring and defense machinery moving.

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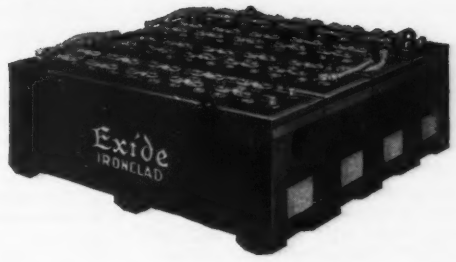

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greater humanitarian importance, also with the creation of an important educational Foundation that will continue to benefit his fellow man for generations to come.

FRANCIS H. BROWNELL

▼ ▼ ▼

LETTERS

Reverberatory Capacity At Noranda

IN THE ARTICLE on "Pyrometallurgy," by Prof. Carle R. Hayward, of Massachusetts Institute of Technology, in *E.&M.J.'s* Diamond Jubilee Number, August, 1941, Professor Hayward was, through an error in this office, made to say: "The little intermittent reverberatories treating 10 to 15 tons grew slowly, then rapidly after fuel oil and pulverized coal became available, up to 800 tons of solid charge per furnace-day." This led W. B. Boggs, superintendent of the Noranda smelter, to write to the Editor saying that Noranda's reverberatories were handling up to 1,600 tons of solid charge per day and that he expected the tonnage to increase still further. His letter was published on page 70 of the October *E.&M.J.* Now we discover that Professor Hayward's article originally contained an additional sentence following the one quoted above. It was as follows: "Gradual improvements since have made it possible to smelt twice this quantity." This sentence was accidentally lopped off the end of the paragraph, after the article had been set in type, in the course of making up the page for the printer.

Apologies are due Professor Hayward for changing the meaning of his statement by deleting part of it, thus making him appear to be unaware of the important progress made at Noranda.

THE EDITOR

Helping the South American Miner

The Editor

DURING THE PAST FIVE MONTHS I have been traveling in South America and have spent some time in Colombia, Bolivia, and Peru examining mines. My work brought me into contact with government officials, foreign diplomats, mining engineers, and many small mine owners, and I was impressed by the general dissatisfaction over the difficulty of obtaining machinery and supplies from the United States.

In his speech of Oct. 7 before the National Foreign Trade Council, Under-Secretary of State Sumner Welles emphasized the acute necessity of maintaining supplies to South America, saying:

"There is likewise the acute problem of the essential import needs of our sister republics of this hemisphere which are largely cut off from European sources of supply. Far too little emphasis, I regret to say, has as yet been placed upon the vital obligation of this country to cooperate to a far greater practical extent than has as yet been the case in assisting to the fullest degree possible our neighbors of the Western Hemisphere in the maintenance of their own national economics in the ever-increasing dislocation to which they are subjected.

"There is also need for additional trade agreements which will help during the emergency and which will assist in establishing a sound foundation for international trade after the war. Your government intends to go forward with this program."

Surely this is the time for both mining circles and manufacturers of mining machinery to adopt a far-sighted policy and so fulfill their obligation to Latin America and at the same time establish trade relations that will continue to be a great asset after the war.

Orders for mining machinery are often held up by "priorities." Though nothing should be allowed to interfere with our national defense program, I believe that the relatively small amount of material required for mining machinery for South America could be spared without hampering in any way our vital defense production.

My observations led me to believe that the smaller mining companies are suffering more than the larger concerns from priority rulings. Now, these small mining companies are for the most part entirely owned by South Americans; and inasmuch as it is well known that the U. S. A. is lending millions of dollars to our sister republics to help in the development of their countries, as well as to compensate them partly for their loss of European trade, it would seem quite logical that our Government take steps to brush away existing difficulties, and help them to obtain the materials and supplies they need to carry on through this difficult period.

JOHN L. MIDDLETON
Casilla 322, Lima, Peru

Converter Smelting

The Editor:

I HAVE READ the letter from F. E. Wing on "Converter Smelting," in the November *E.&M.J.* To my personal knowledge the smelting of ores in converters has been common practice for over 30 years. This was true both under acid, as well as basic, converting. In the former the ore was first charged into converters by means of "boats" from overhead cranes, when the acid linings were barren siliceous rock. In 1908 at Great Falls, Mont., we gradually replaced the barren linings with cupreous material—Butte copper ore and copper precipitates. Charging of ores through the converter mouths continued at all times.

Direct smelting of fine sulphide concentrates was tried for a period during 1907 to 1909 by blowing this material through the tuyeres. Dr. Ricketts improved on our practice by blowing fine concentrates through one tuyere by a pipe arrangement, which could be quickly connected and disconnected.

With the introduction of basic converting this practice was discontinued, and ores and sulphides were charged only through the converter mouths, and it is the practice today, here and abroad. Knudsen introduced his smelting converter in Norway about 30 years ago, and his method was tried out at the United Verde, but the results apparently did not justify its continuance.

At Great Falls we carried on many tests to determine just how much sulphide material could be smelted in converters. We found that with the grade of sulphides we had from our concentrator, there was a gradual increase in the grade of matte in the converter, until the white-metal stage was reached, when the charge was blown to copper.

The operation worked the best with coarse concentrates from our coarse jigs. Using fine concentrates, there was a heavy flue-dust loss, so that Mr. Wing's suggestion of sintering or agglomerating fines is a good one.

A reference to a paper by Archer E. Wheeler and myself, published in the *Transactions* of the A.I.M.E., Montana Volume, 1913, shows the amount of ores smelted in detail. Reference is also made to more recent articles in the same *Transactions*, by George E. Beavers and F. J. Longworth, on smelting concentrates in converters at the plant of the Tennessee Copper Co. (Vol. 106, page 149, and Vol. 71, page 969, 1933 and 1925).

Introduction of selective flotation concentration, and the production of concentrates high in copper, and the cheapness of smelting in large reverberatories, have made it hardly worth the effort to use the converter as the smelting unit. With fuel ratios of 10 percent or even less, the actual money saving is not great. This saving would have to be enough to compensate for the greater cost of high-pressure converter air, and higher cost of getting the ore to the converter. To this would have to be added the agglomerating or sintering cost.

It seems that the possibility of smelting copper sulphides, from the raw state to finished copper, offers more chance of being accomplished without the use of extraneous fuel in a specially designed reverberatory furnace, having ample combustion space, by blowing them into the furnace in a fine state. Flotation concentration produces them in this fine condition. Although there is sufficient heat generated by the oxidation, there would be a slight deficiency, due to radiation and loss of heat in the waste gases. By preheating the secondary air to the furnace with the waste gases or by slightly enriching the air with oxygen, or both, the direct smelting of sulphide concentrates can be accomplished without fuel.

Specially designed nozzles would have to be used, to keep the sulphide particles in suspension until oxidation is complete, or there would be only a partial oxidation. We conducted tests in this procedure at Great Falls in 1916, to demonstrate sufficiently that smelting without fuel from the raw sulphides to copper can be accomplished.

Direct smelting of sulphides in a converter can be done, but the reverberatory can do it more cheaply, even though extraneous fuel is used. To smelt 1,500 tons in converters would be a tremendous daily problem in handling alone. In some plants a single reverberatory does this.

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Washington, D. C.

What Metal Statistics Show

WAR BY JAPAN on Pacific possessions of the United States on Dec. 7 brought government officials and mine operators together, studying the price situation in copper and lead, in the conviction that mines must increase output. Metals are needed now for war. In World War I domestic copper producers in 1916 obtained a higher yield of copper per ton of ore and received an average of 27.2c. a pound for copper and produced a volume of metal less than that expected to be produced in 1941 at 12c.; in spite of a much lower yield of copper per ton of mined ore. Public hearings were planned to be held in Washington in December to obtain data for the increase of copper output.

Lead will be discussed Jan. 5, as Leon Henderson head of OPA has agreed to confer with representatives of the industry to determine whether the price of lead should be increased. The conference was called by Representative White, chairman of the authority considering the price situation in lead.

Control regulations affecting distribution and use of copper were a factor during October in holding down deliveries and bringing about a modest rise in stocks of duty-free copper. The statistics of the Copper Institute for last month showed stocks of refined copper on Oct. 31 of 67,260 tons, which compares with 63,670 tons a month previous and 142,772 tons at the beginning of the year.

The summary of copper statistics for September and October, in tons, follows:

	Sept.	Oct.
Production:		
Crude	(a) 81,839	85,546
Refined	(a) 81,553	86,617
Deliveries, refined:		
Production:	Sept.	Oct.
Domestic	(a)121,021	121,313
Export	.....	144
Totals	(a)121,021	121,457
Stock at end (b)	63,670	67,260

(a) Corrected. (b) Duty-free copper at refineries, on consignment, and in Exchange warehouses, but not including consumers' stocks at their plants or warehouses.

Imports of copper into the United States during September, with comparable figures for the same month last year, in tons:

	September	
Copper in:	1940	1941
Ore and concentrate	5,323	4,485
Matte, etc.	7,426	568
Unrefined	31,416	25,431
Refined	7,320	34,915
Scrap	122	55
Totals	51,607	65,454

▶ LEAD—Statistics covering lead refined in this country for the months of September and October, in tons, according to the American Bureau of Metal Statistics, follow:

	Sept.	Oct.
Stock at beginning	15,330	13,148
Production:		
From domestic ore	41,373	37,221
Secondary and foreign	3,530	3,906
Totals	44,903	41,127
Domestic shipments	47,093	43,537
Stock at end	13,148	10,735

"Present ore production is about the maximum possible at the current lead price," Felix E. Wormser, secretary of the Lead Industries Association, told a House committee headed by Representative White (Idaho). "Mine operators," he said, "are experiencing difficulties in keeping workers at present rates of pay and may have to increase wages, and therefore ore prices must be raised to maintain the present output rate." The committee has been named to look into the price situation in lead. Sentiment in the Rocky Mountain states is strong for raising the price to stimulate production of lead.

▶ ZINC—The domestic zinc statistics (all grades) for October and November, according to the American Zinc Institute, follow:

Production of Prime Western and other grades for November, follow:

	Oct.	Nov.
Stock at beginning	19,427	21,594
Production	75,980	74,710
Total	95,407	96,304
Production daily rate	2,451	2,490
Shipments:		
Domestic	61,594	60,913
Foreign (drawback duty)	12,219	12,209
Total shipments	73,813	73,122
Unfilled orders	62,559	84,101
Stock at end	21,594	23,182
Production by Grades		
Prime Western	29,730	29,431
Select	1,456	1,417
Brass Special	7,077	4,364
Intermediate	3,596	4,695
High Grade	18,043	16,399
Special High Grade	16,078	18,404
Total	75,980	74,710

In reviewing the zinc situation at the meeting of the New York Section of the American Institute of Mining & Metallurgical Engineers, on Nov. 27, Howard I. Young, president of the American Zinc, Lead & Smelting Co., said it is reasonable to assume, in view of the possible diversion of steel to other uses and the trend toward substitute coatings, that the reduction in the use of zinc by the galvanizing industry in 1942 will be adequate to account for a difference of 30,000 tons between consumption plus exports and the available supply.

Mr. Young presented the following statistics of the needs of consumers, plus exports, for 1941 and adjusted consumption in 1942, in tons:

	1941	(a)1942
Galvanizing	312,000	312,000
Brass mills	240,000	360,000
Die casting	120,000	100,000
Rolling mills	84,000	67,000
Oxide plants	18,000	18,000
Other uses	12,000	12,000
	786,000	869,000
Estimated exports	150,000	150,000
Total needs	936,000	1,019,000

(a) Estimated consumption based on adjustments to balance production and consumption.

▶ ARSENIC—Production of crude arsenic in the United States during September was 2,906 tons, making a total of 30,367 tons for the first nine months of 1941. Production during all of 1940 was 35,573 tons.

Imports of white arsenic during the first eight months of 1941 amounted to 6,499 tons, compared with 8,362 tons in the same period last year.

▶ MOLYBDENUM—Production of molybdenum concentrate during September was 3,816,600 lb., against 3,780,300 lb. in August, according to the Bureau of Mines. The statistics for August and September, in pounds, follow:

	Aug.	Sept.
Stocks at beginning	18,581,100	18,096,700
Production (a)	3,780,300	3,816,600
Shipments:		
Domestic	2,216,400	3,002,800
Export	2,048,300	91,100
Totals	4,264,700	3,093,900
Stock at end	18,096,700	18,819,400

(a) Includes smelter salts produced from complex ores.

▶ QUICKSILVER—Domestic production of quicksilver in September reached 4,200 flasks, against 4,100 flasks in August and a monthly average of 3,579 flasks for the first nine months of 1941, the latest report on the metal by the Bureau of Mines shows.

Quicksilver statistics for August and September, in flasks of 76 lb., follow:

	Aug.	Sept.
Production	4,100	4,200
Imports	500	(a)
Exports	318	(a)
Consumption	3,600	3,700
Stocks (b)	12,157	12,716

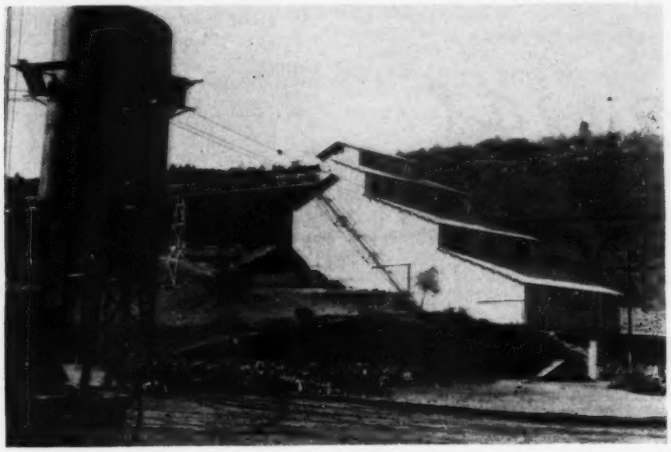
(a) Not yet available. (b) In hands of consumers, dealers, and producers reporting to the Bureau of Mines.

E & M J Weighted Index of Non-Ferrous Metal Prices

(100) is composite for 1922–3–4)

1929	110.33	1935	74.66
1930	82.87	1936	73.45
1931	60.20	1937	90.86
1932	48.26	1938	73.67
1933	59.79	1939	77.71
1934	69.59	1940	79.22

	1939	1940	1941
January	77.12	80.85	82.48
February	76.89	77.23	82.76
March	77.00	77.94	83.27
April	74.79	77.19	83.48
May	73.61	77.68	83.48
June	73.17	78.85	83.51
July	71.95	76.72	83.60
August	73.88	76.71	83.39
September	83.07	79.40	83.34
October	84.25	82.49	83.84
November	83.80	83.14	84.32
December	82.89	82.42	....



View of new 350-ton flotation mill erected recently by Madre de Oro Gold Mines, Inc., at the Church mine, near El Dorado, El Dorado County, Calif.

CALIFORNIA

Walker Copper Mine Suspends Operations

Present price of the metal too low to maintain production in face of rising operating costs—Oro Fino Consolidated Gold Mines begins deeper shaft sinking

▶ Operations have been suspended at the Walker mine, California's largest copper producer, at Walkermine, Plumas County, because of the current low price of copper. The mine is controlled by the International Smelting & Refining Co., of Salt Lake City, subsidiary of Anaconda Copper Mining Co., and employed up to 500 men when working at capacity. The concentrator is equipped to handle 1,500 tons of low-grade ore a day, and production in 1940 amounted to 10,524,345 lb. copper, 237,891 oz. silver, and 14,176 oz. gold. Operations were directed by H. M. Hartmann, manager.

▶ After completion of an extensive diamond-drill campaign, Oro Fino Consolidated Gold Mines has started to deepen the 800-ft. incline shaft at its property at Ophir, Placer County, an additional 400 ft. The mill is currently operated on a one-shift basis, handling ore mined from upper levels. J. C. Kemp van Ee, general manager, is in charge of operations.

▶ Rustless Mining Corp., C. E. Tuttle, president, a subsidiary of the Rustless Iron & Steel Corporation, Baltimore, Maryland, has started work on a 200-ton concentrator designed by Southwestern Engineering Co., of Los Angeles, to treat chrome ores to be mined by open-pit methods at the Grey Eagle mine, at Chrome, Glenn County. Transportation of ore from the mines to the mill will be effected by an aërial tram about 2 miles long. Actual production is expected to start early in February. C. E. Osborne, general superintendent, directs operations both at the Grey Eagle holdings and the chrome mining operations conducted by the company in El Dorado County. Robert H. Sayre is vice president in charge of production, with office at Sacramento, and H. F. Byram is vice president in charge of development, with headquarters at Grants Pass, Oregon. The company has chrome properties in California, Oregon, and Washington.

▶ Lobicasa Co., of Sacramento, has moved one of its dragline dredges working within the flood control dam of the city of Stockton, on the Calaveras River near Valley Springs, to the Jamison Creek placers, near Quincy, Plumas County. A second unit will be moved to the same holdings soon.

▶ At the Midnight mine, near Nevada City, Nevada County, reopened recently by local interests, lateral work is in progress, following dewatering of the 180-ft. incline shaft. Nine men are employed under the direction of L. C. Smith, superintendent.

▶ Lincoln Gold Dredging Co. recently acquired by lease the old Niagara mine and adjoining mineral holdings about 6 miles west of French Gulch, Shasta County. Examination and exploration work has proved satisfactory to the extent that construction of a 100-ton amalgamation-flotation mill was decided on, with foundation work being started on Sept. 20. The plant will handle semioxidized porphyry and slate quartz ores contained in the old dumps on the properties for several months pending completion of underground and surface mining preparations, and milling operations are scheduled to commence early in December. Tests made on the ore indicate that a high percentage of the gold can be recovered by amalgamation; values existing in pyrite and galena, along with free gold, will require flotation treatment. The mill flowsheet follows closely that of the newer mills installed in California, and equipment in course of erection includes a jaw crusher, ball mill with classifier, jig, barrel amalgamator, flotation cells, filter, and cleaner table. Ore mined at different workings will be transported to the mill by dump trucks. Paul A. Bundy, of Nevada City, is superintendent in charge of design and construction of the mill and surface plant layout, and eleven men are employed. The company is headed by W. K. Jansen, E. M. Clark, and associates.

▶ Construction of a mill with a maximum capacity of 350 tons and a surface plant is nearing completion at the Church mine, near El Dorado, El Dorado County, operated by Madre de Oro Gold Mines, Inc. Mining machinery erected recently includes a 200-hp. double-drum hoist and a 750 cu.ft. compressor. The mine is opened by the Church shaft, 1,200 ft. deep and having ten levels. Underground development is proceeding steadily, with production scheduled to start at the end of the year. Forrest Riley is president of the company, Mrs. Hazel Riley Mitchell, secretary-treasurer, and W. J. Loring, managing engineer.

▶ Douglas Mercury Co. has started operation at the Manzanita quicksilver mine, near Wilbur Hot Springs, Colusa County, following installation of a new treatment plant. B. C. Austin, San Francisco consulting engineer, is manager.

▶ Dredging operations have been started at the old Malakoff hydraulic holdings near North Bloomfield, Nevada County, by the Innis Dredging Co. after reassembly of the dragline dredge formerly working on Deer Creek below Nevada City. A. B. Innis is superintendent.

Dragline dredge unit of River Pine Dredging Co., operating on Consumnes River not far below the old Nashville mine, near Nashville, El Dorado County, Calif.

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4. Z4—Sodium Ethyl Xanthate
5. Z5*—Potassium Amyl Xanthate
6. Z6*—Potassium Pentasol Amyl Xanthate
8. Z8—Potassium Secondary Butyl Xanthate
9. Z9—Potassium Isopropyl Xanthate
10. Z10—Potassium Hexyl Xanthate

**From Sharples Amyl Alcohol*

At the Golconda division of the Nevada-Massachusetts Co., in Nevada, this new plant is treating tungsten ore, from the company's mine about three miles away, where paving breakers are employed in mining. The capacity of the plant will be increased from 100 to 200 tons per day.

NEVADA

Drilling Under Way on Old Copper Claims

International Smelting & Refining Co. investigating Guild—Adams-Herrin property in Lyon County—Nevada Massachusetts Co. to double capacity of its tungsten plant

▶ Diamond drilling is in progress under contract for the International Smelting & Refining Co. on copper claims two miles west of Yerington, Lyon County, known as the Guild-Adams-Herrin property, from which the long-defunct Empire-Nevada Copper Mining & Smelting Co. shipped some $500,000 worth of high-grade copper ore, mined at shallow levels. In charge at the property is Robert S. Moehlman, of Anaconda's geological department at Butte. International, a subsidiary of Anaconda Copper Mining Co., is completing a 350-ton flotation mill at the Copper Canyon mine of the Copper Canyon Mining Co. in Lander County, developing this mine and the company's Copper Basin properties.

▶ Capacity of a 100-ton tungsten mill operated by the Golconda Syndicate, subsidiary of the Nevada-Massachusetts Co., near Golconda, southeastern Humboldt County, is to be doubled at once. The scheelite ore is mined by power shovel from a flat terrain and trucked to the mill. The Nevada-Massachusetts operation near Mill City, Pershing County, until recently the largest United States producer of tungsten, treats 260 tons of scheelite per day and 1,000 tons per day of tailing that has accumulated from the primary treatment plant. Late reports stated an RFC advance had been authorized to increase capacity of this mill.

▶ Provision has been made to increase substantially the production of dumortierite from the only known important deposit of the mineral, in western foothills of the Humboldt range in Pershing County, owned by the Champion Sillimanite, Inc., wholly-owned subsidiary of the Champion Spark Plug Co., of Detroit. Used in spark-plug cores, dumortierite is mixed with sillimanite from the company's deposits high on the White Mountains in Inyo County, Calif.

▶ Lead-silver concentrate is being shipped from the remodeled flotation mill at the Nevada-Belmont mine in the Hamilton or White Pine district, a carload having been shipped recently to the A. S. & R. lead smelter at Garfield, Utah. The mine, outmoded mill, and two-mile aërial tramway were purchased some months ago from the Tonopah Belmont Development Co. by A. A. DeMelik, mining engineer of New York, who lately was said to have sold the property to Byron F. Snyder, of Lauderdale, Fla. Edward R. Wagner, of Ely, has been appointed manager.

▶ Prospect drilling, conducted by contractors for the Freeport Sulphur Co. on the old Majuba Hill copper mine in the Antelope district, Pershing County, has been discontinued for the present, but results of underground work in the vicinity of an isolated shoot of tin ore have encouraged further exploration. A. A. Gustafson, engineer in charge of Western operations for the Freeport Sulphur Co., with offices in Reno, said low-grade cassiterite had been found by prospect work over a considerable area. The shoot or pipe of high-grade cassiterite occurs in a fissure that produced several thousand tons of 12 percent copper ore in earlier work. In view of the improved outlook, Representative James G. Scrugham, of Nevada, has requested the U. S. Bureau of Mines to renew its investigation of the tin occurrence.

▶ Sinking of a new deep shaft was started on Nov. 15 by the Eureka Corporation, Ltd., of Toronto, on the Richmond-Eureka property on Ruby Hill, two miles west of Eureka. Site of the shaft is 1,400 ft. north of the Locan shaft, main opening of the old mines which produced some $80,000,000 in lead, silver, and gold. The shaft is designed to reach the ore zone north of the Ruby fault and at a depth of 2,500 to 3,000 ft., diamond drills having established the presence of ore near this depth. William Sharp is manager for the Canadian interests, who will discontinue work on their Oswego gold property in nearby Lecret canyon, having shipped available ore to the Kennecott smelter at McGill.

▶ Offices, warehouses, and other structures to house a large number of workers are being built at the site of the Basic Magnesium, Inc., plant midway between Boulder Dam and Los Vegas, in Clark County. The magnesium plant, principal unit of a $63,000,000 project that includes calcining furnace installation at the magnesite deposit, 300 miles to the north, mining and delivery of a large tonnage of salt, together with power and water supply from Boulder dam, will give employment to some 4,000 men. Contract for construction of the main plant, financed by the Defense Plants Corp., subsidiary of the RFC, was awarded to the McNeil Construction Co., of Los Angeles; a $3,000,000 contract for the magnesium oxide plant at the deposit in Gabbs Valley to the MacDonald Engineering Co., of Chicago, and another involving $6,100,000 for power and water lines to the main plant to the J. M. Montgomery Co., a combination of seven southern California firms.

ARIZONA

Castle Dome Mining Co. Gets RFC Loan

Miami Copper will develop and bring its subsidiary into production with $9,000,000 financing program

▶ The Defense Plant Corp., RFC subsidiary, has arranged a $9,000,000 financing program to provide facilities for producing 23,000 tons of copper annually, under a lease agreement, at the property of the Castle Dome Copper Co., Miami, Ariz. Mining will be by open-pit method and a concentrator, having a capacity of treating 10,000 tons of ore daily, will be constructed and be ready about March, 1943. Castle Dome is a subsidiary of Miami Copper Co. Arno S. Winther, Box 100, Miami, Ariz., is general manager.

▶ The American Smelting & Refining Co. is producing about 80 tons of lead-zinc ore daily from the Octave mine, near Octave. About 85 men are employed under the direction of LaMont West, superintendent.

▶ About 85 tons of gold ore are being shipped daily from the Golden Door mine, 18 miles northwest of Chloride, in the River range, to the Producers, Inc., mill 7 miles west of Chloride. The values are mostly in gold. Keene St. Charles, of Kingman, is owner of the property and Dye and Bathrick are operating the property under a sublease from J. A. Potter, also of Kingman. Mining at present is done mostly with a dragline, with trucks hauling the ore to the Producers mill.

▶ L. Espey and Thomas Wheelock, representing a Chicago firm, are operating a dry washing plant recovering gold from gravels two miles south of Quijotoa. They have been experimenting with the dry process for the past two or three months, and the present equipment is handling about 300 cu.yd. of gravel daily. About 15 men are carried on the payroll.

▶ The Vivian Mining Co., Inc., operating near Oatman, Mohave Co., has completed overhauling of its 50-ton plant and is now making plans to double capacity. Ore developments of the company's property have been satisfactory and also

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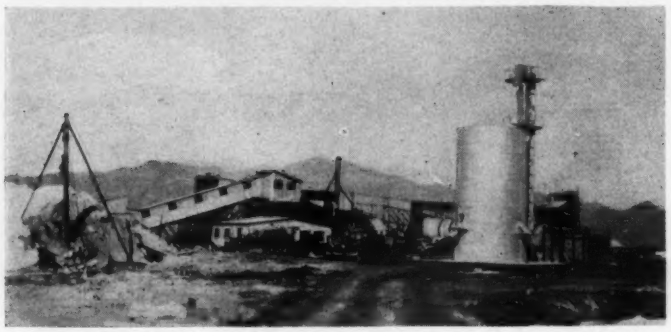

Bonneville, Ltd., the only producer of potash in Utah, near Wendover, Tooele County, has increased its output of salts from its operation involving flotation and solar evaporation to purify the brine from a marsh through successive stages of concentration and crystallization

▶ARIZONA

custom ores from elsewhere in the district have added to the necessity for enlarging the capacity of the milling plant. A. E. Laitem, of Oatman, is manager of the property. A. E. Kern, of 110 Sutter St., San Francisco, is president of the company.

▶ Ten men are reported employed by the Canyon Mines, Inc., Hereford, Ariz., on its property in the Huachuca Mountains in Cochise County. The ore is said to contain gold, silver, and tungsten. Gold, Silver & Tungsten, Inc., of Boulder, Colo., is owner of the property. Frank Fields of Mammoth, Ariz., is manager.

▶ Empire Mining Co., owners of the Empire mine, 11 miles north of Parker, in Yuma County, is under lease to A. W. Clapp, W. W. Harrit and F. W. Royer, of Parker. Clapp formerly operated in Mohave County. Shipments are just starting. The gold and copper property is developed by a 600-ft. shaft, but present operations are mostly on the 200 level, which is opened by an adit tunnel.

▶ Development work accompanied by ore shipments is going forward at the old King of Secret Pass mine, in Secret Pass, 20 miles west of Kingman, in Mohave County. J. H. Hoffman, of Kingman, is managing the property and R. C. Ferguson is superintendent. The property is owned by Walter M. Harrison.

▶ The Pioneer mine, just southwest of Oatman, which was worked by a large number of leasers for several years, is again in production and gold ore is being shipped to a custom mill. George Moser, a supervisor of Mohave County, controls the property.

▶ A number of parallel veins have been opened on the Alamo group of claims a few miles southwest of Artillery Peak in southern Mohave County and considerable cinnabar ore has been uncovered. This is the first showing of quicksilver found in that area. The property is owned by Mrs. Rachel Thompson, of Aguila.

▶ P. D. McIntyre, of Blythe, Calif. who owns a vermiculite property 10 miles west of Quartzsite, Ariz., is arranging to fill contract for delivery of the ore. It will be hauled by truck to a shipping point on the Atchison, Topeka and Santa Fe Railroad.

▶ A carload of gold ore is being shipped weekly from the Little Giant mine, 10 miles north of Aguila. A. MacFarlane is owner of the property and the work is being done by leasers. Some copper is reported present in the ore.

▶ Bids have been received at the Window Rock Indian Agency on the Navajo Indian Reservation for the leasing of a copper property 20 miles from the trading post known as The Gap. Bids are required to offer a cash bonus, annual rental, and stipulated royalties. The leaser is also required to use Navajo Indian labor for all work not necessitating skill. Appraisements are made by the United States Geological Survey.

▶ Because of cavings in a glory hole, operations at the Tin Cup mine, in the Secret Pass section of Mohave County 20 miles west from Kingman, the property has been shut down. C. D. Hewen, of Kingman, who was in charge of operations, has given up his lease. Other parties are now negotiating for the property.

▶ The Rainbow mine, a few miles east of Chloride, has been leased by Henry Hall and his two sons, Kenneth and Ray. The property has been an intermittent producer of gold, silver, lead and zinc ore for many years.

UTAH

Iron and Antimony Deposits Being Investigated

Federal Bureau of Mines studying some areas in Washington County—North Lily Mining Co. driving to explore Tintic Bullion and Eureka Bullion claims

▶ Utah deposits of coal, iron, and antimony are being tested by the Federal Bureau of Mines under the direction of Paul T. Allsman, engineer in charge of the mining division office at the intermountain Station, Salt Lake City. Tests of iron deposits are being made on the Milner properties, in Bull Valley, Washington County. Seven miles of road have been built to the site of operations and 30 men are engaged in trenching, sampling, and drilling. A. B. Needham is project engineer. East of Mount Pleasant, diamond drilling of coal beds believed to have value for coking is under way. Five miles from Antimony and 34 miles southeast of Marysvale, an antimony deposit is to be prospected. W. M. Traver, Jr., has charge of the trenching and drilling.

▶ Bonneville, Ltd., of Salt Lake City, Utah's only potash producer, has increased production at its Wendover plant, near the Utah-Nevada line, to 100 tons of potash salts daily. At the Salduro brine marsh 3 miles from the plant the company utilizes solar evaporation to purify the brine through successive concentration and crystallization. The crystalline mixture of potassium and sodium chlorides is trucked to the flotation plant, enough of the salts being trucked in one shift to keep the mill running three shifts around the clock. About 35 men are employed. The company has overcome technical difficulties to work out a process which produces high-grade muriate of potash around 98 percent pure from the crystallized brine running 20 to 30 percent potash. From the flotation plant the potash salts go through a rotary drier and into a large storage tank (center photo) from which they are loaded into covered trucks and hauled 4 miles to Wendover for shipment.

▶ North Lily Mining Co. is advancing the west drift on the 1,500-level 10 ft. a day to explore the Tintic Bullion and the Eureka Bullion claims along the Eureka Lilly fault. Similar work is being done on the 1,200- and the 1,350-ft. levels.

▶ Ophir Treasure Mining Co. recently shipped a 40-ton carload of ore from its Dry Canyon property. The company is developing its Kearsarge property through the Mono tunnel. C. D. Clegg has reopened the tunnel in the Silver King Extension property, northeast of Park City. New machinery is to be installed and the tunnel advanced 100 ft., where a shaft will be sunk to the Park City formation.

▶ Silver King Coaliton Mines reported a net profit of $131,995 for the third quarter of 1941 before depletion and Federal taxes, compared with net earnings of $91,728 for the same quarter in 1940.

▶ Operations have been resumed at the Georgia Lynn mine, near Delle, Tooele County, by W. W. Choate. The Dragon Consolidated Mining Co., at Eureka, recently completed the shipment of 300 tons of halloysite clay for filtering purposes to a Los Angeles oil firm.

▶ The Ophir Midland Mining Co., in the Ophir district, Tooele County, is driving the Gem tunnel to reach the extension of the limestone productive in the adjoining Hidden Treasure mine.

▶ A showing of scheelite ore is being developed by the Little May Mining Co. in Tintic, according to John Matson, president.

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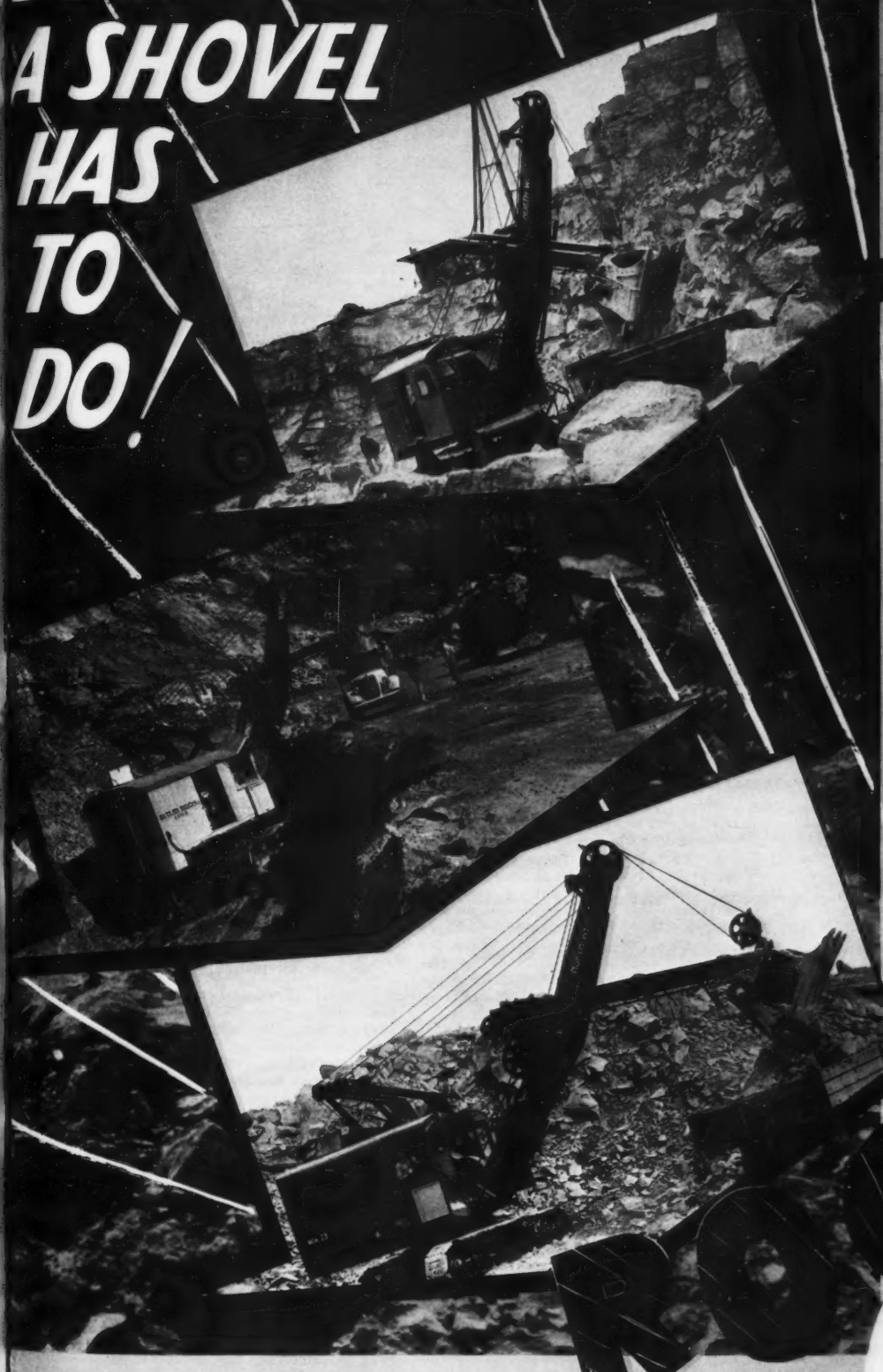

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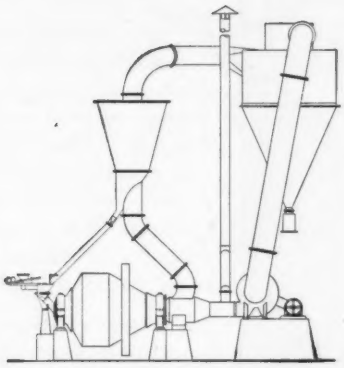

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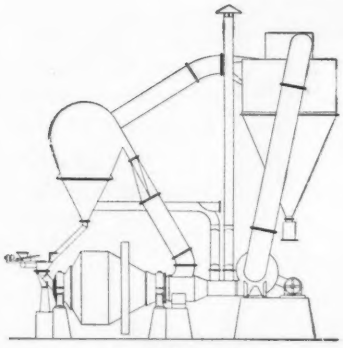

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The Cresson mine, Cripple Creek, Colo., will eventually be connected with the Carlton tunnel upon completion of a 4,000-ft. lateral, permitting dewatering of the lower levels of this mine. Leasors are operating the upper levels.

COLORADO

Mineral Resource Survey Undertaken

Information and data on possibilities in Gilpin County being collected to determine deposits of critical and strategic minerals—Shaft sinking continues at Ajax mine

▶ The Mineral Research Board, together with the State Mining Association and the Metal Mining Fund, under the direction of Edward D. Dickerman and J. A. Lintner, are making a survey in Gilpin County to determine mineral resources, particularly with regard to critical and strategic minerals. Names of owners of patented mining claims are listed and questionnaires mailed to them, requesting they list the various minerals and estimated production of such: or state why such claims cannot be worked at this time. This survey is made in hope that the government will be influenced, and lend assistance to the small operators. W. E. Scott, Jr., member of the Mineral Research Board, was the instigator of this plan.

▶ Two shifts are now engaged in sinking the Ajax shaft, and already they are some 70 ft. below the 26th level, and, according to Superintendent Charles Carlton, they will continue another 100 ft. A new electric locomotive built by Master Mechanic Guy Rorabaugh has been installed on the 24th level, replacing the mule. Two more locomotives will be built at the Cresson shops for other Carlton mines. The locomotives will greatly speed production at their properties; and they are built similar to those used in the construction of the Carlton tunnel, only smaller.

▶ The following report has been issued by K. M. Ohlander. president of the Gold Mines Consolidated, Idaho Springs: "The oreshoot on the 400-ft. level of the Donna Juanita mine is approximately 500 ft. long, with ore continuing in both drifts. Two raises 200 ft. high have ore in them all the way, with an average width of about 3 ft. and average value of $6.50 per ton. We should be able to count upon at least one year's supply of ore mining at the rate of 50 tons daily. Further the possibilities of ore in other veins besides the one we are working on are good. The mine is producing more ore than the mill can handle. Arrangements are being made for new milling facilities." Gross production for November was around $10,000.

▶ The surface plant of the Shattuck Chemical Co., at Boulder, that was recently destroyed by fire, has been rebuilt; and a new electric-driven compressor and hoist installed, replacing the old gasoline-driven units. A large body of low-grade tungsten ore is being mined from the 100-ft. level, and development continues in ore on the 200 level.

▶ J. Price Briscoe, president of the Conqueror Operating Co., Idaho Springs, is planning a three-compartment 500-ft. winze. The ore will be treated in the Humboldt mill, which Mr. Briscoe is now operating part time on custom ore. Otto Carstens is mill foreman and Mike Evans mine foreman.

▶ George H. Teal, of Boulder, is mining and milling some 15 or more tons of tungsten ore daily, from his two properties, the Princess-Gold Coin group and the Dorothy-Katie. The mill is of 30 tons' capacity and is under the direction of Walter Swanson. J. B. Giggey is mine superintendent.

▶ The General Chemical Co., of New York, is contemplating reopening its fluorspar property at Jamestown. The Valmont mill will be under the supervision of E. H. Brooks and Mr. Jefferson will be in charge of all operations. The fluorspar mines were purchased two years ago, and have remained idle since, as the General Chemical Co., has been milling fluorspar near Deming, N. M.

▶ Treasure Mountain Gold Mining Co.'s mill, at Silverton, designed and built by Fred A. Brinker, has been placed in operation. This new mill, is at the portal of the Sandiago tunnel, in Picayune Gulch. This first unit was designed as a pilot plan to make test recoveries on the ores of the several mines. Con-

tinuation of the Sandiago tunnel to the original objective—the Golden Fleece and Scotia veins—is planned. An additional 500 ft. of tunnel work will permit mining 800 ft. deeper on the veins and delivery of ore to the mill at the portal of the tunnel.

▶ Roy Peterson, in charge of the Sunset No. 2, south of the Anna C, at Boulder, reports having cut a fine vein of tungsten ore. This shaft was sunk as the result of diamond-drill operations.

▶ During the clean-up work on the lower levels of the Portland mine, as a result of the New Carlton drainage tunnel, the Portland mine is mining a large body of good ore on the Trail Basalt dike from the 17th level. Plans call for connecting the drift with the 14th level of the Cresson mine.

▶ Walter Graskewiez, in charge of operations at the Montyne Schnell lease on Poorman Hill, Boulder, has just completed installing a new 440-cu.ft. compressor with a 60-hp. motor, a 30-hp. two-stage pump, and an electric hoist. Mining and development continue, with shaft sinking planned later.

IDAHO

Future Wage Rise Based On Higher Lead Price

Miners rate of pay on Jan. 16 agreed to at $7.25 a shift plus 25c. per shift increase for each one-half cent increase in the lead price above 6.35c.

▶ Federal Mining & Smelting Co. has signed an agreement with the Mullan local Miners Union No. 9, and the Kellogg local, No. 18, both affiliates of the CIO, governing labor conditions at the Morning and Page mines. Wages are fixed at $7 per shift until Jan. 15, 1942, plus 25¢ per shift for each one-half cent increase in the price of lead above 5.85¢ per pound. Commencing Jan. 16, 1942, the miner's wage rate is fixed at $7.25 a shift, plus 25¢ per shift increase for each one-half cent increase in the lead price above 6.35¢ per pound. The local unions are recognized as bargaining agents, but the company does not recognize a union or closed shop. Employees who have been in the company's service for one continuous year are entitled to a seven-day vacation period with pay. The agreement was signed by representatives of the Mullan Miners Union and Kellogg local, both affiliates of the CIO. Representing the Federal company was H. G. Washburn, general manager, and J. E. Berg, general superintendent. The Mullan union was represented by J. C. Burrill, Rayno Kangas, L. Gustafson, and C. J. Powers. Representatives from the Kellogg local were T. W. Liepold, A. McKinley, and Wayne Lewis. Similar contracts have been agreed to at the Tamarack & Custer, Sherman, and Dayrock mines of the Day organization. Bunker Hill, Sunshine, Coeur d'Alene Mines, and Hecla all have separate contracts.

▶ Telegrams from the OPM suggesting working six days a week and seven if possible in order to speed up the output

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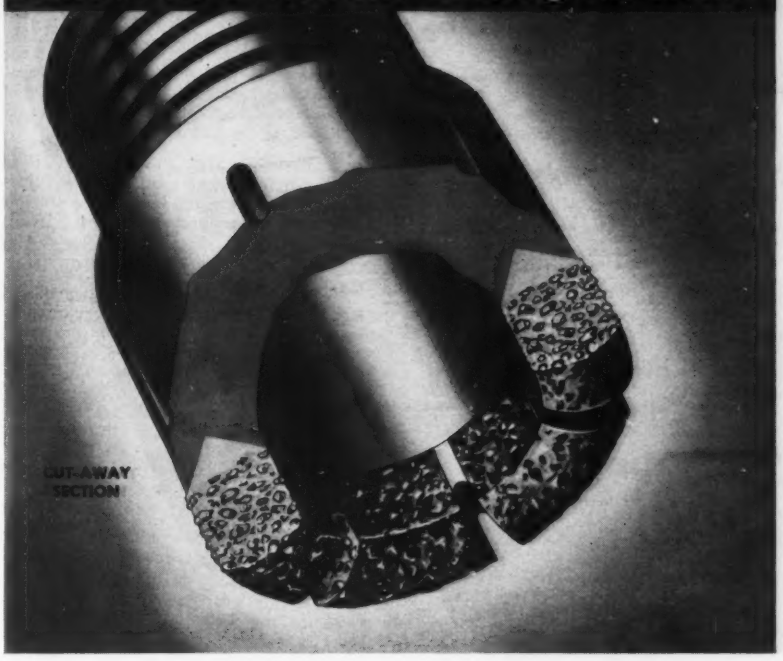

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▶IDAHO

of lead did not receive very enthusiastic reception in the Coeur d'Alene mining district. Idaho mining men claim that lead is "undernourished" and is the "forgotten" metal. They assert that the lead price is not adequate to the zinc price and admit that both metals should be advanced to meet the increased costs of labor, taxes and supplies.

▶ Highland-Surprise mine, in the Pine Creek district, has completed the Surprise shaft to the 300 level and a station is now being cut preparatory to drifting on the vein. If the lead content of the zinc-lead ore increases, the company may resume milling operations.

▶ The Coeur d'Alene Mines Corp., a new and growing producer of copper, antimony, and silver, has declared a dividend of 8¢ per share, payable Dec. 23. This is the company's initial dividend. It will total $231,603.12 and be distributed to 2,300 stockholders. Dr. H. C. Mowery is president, Dan Leary vice president and manager; L. Randall secretary and treasurer, James Doyle mine superintendent, and Robert Baily, chief engineer. The company office is Wallace, Idaho.

▶ The old Banner silver mine, in the Boise Basin, near Idaho City, is being reopened by Greenhalj brothers, who have leased the property from Vivian Thorne. About ten men are employed sinking a shaft 200 ft. to reach the old workings. The Banner mine was located first in the '60s and at one time employed 200 men, with a production of $1,500,000.

▶ The Callahan Zinc-Lead Co., operating the Deadwood mine, near Cascade, is now repairing the old milling plant to treat the silver-lead-zinc ore opened in recent development work.

▶ Sunshine Consolidated Mining Co. reports improved conditions in drifting on its silver vein from the 3,100-level of the Sunshine shaft. The drift has been advanced 400 ft. and though the entire drift is mineralized, it is reported not sufficiently rich for commercial ore.

▶ Atlas Mining Co., at Mullan, being developed by the Hecla company, has opened a promising showing of galena ore on the 800 level in a 1,000-foot drift east from the station. Good lead ore has been showing for the last 40 ft. from small streaks up to widths of 18 in. Ralph Neyman is engineer in charge.

▶ Mohawk Mining Co. is starting work in the Elk Creek district near Kellogg on a group of eight claims which have a good surface showing of silver-lead ore. Albert M. Nash, president, Clarksfork, is in charge of operations.

▶ The Zola Mines, at Summit Flat, in Boise County, have been sold to L. E. Eck and Charles Hillock, of Boise, by Lee Bunch, under a bond-lease agreement calling for a total payment of $52,000.

▶ Lucky Friday mine, at Mullan, has started shipments to the Golconda mill from the 300 level, where the vein has streaks of high-grade silver ore. A new electric hoist is being installed to sink the main shaft another 200 ft. Development is being financed by the Golconda Lead Mines Co., of which A. H. Featherstone is president and manager, with office at 504 Bank St., Wallace, Idaho.

▶ Idaho-Leadville Mining Co. is diamond drilling on its group of claims on the east fork of Big Creek, near Kellogg. James L. Leonard is managing engineer in charge of the operations and reports the presence of several veins of siderite and quartz, characteristic vein filling of the silver veins of the district.

WASHINGTON

Antimony Queen Mine Being Developed

Properties in Okanagan County will ship to smelter in Laredo, Texas—Tungsten ore being explored in the Red Top mine.

▶ William Oldfield has taken a lease on the Antimony Queen property, in the Methow district, and is preparing to make shipments to the smelter at Laredo, Texas. The property is owned by H. B. Johnston and C. E. Barr, of Seattle, and George Gibson and Vernon Lematte, of Twisp. Antimony ore from the Lucky Knock mine, near Tonasket, Okanagan County, will be shipped in the same car.

▶ Scheelite ore has been discovered and is being explored in the Red Top mine, near the international boundary line, in Stevens County. It was found in an old winze which was intersected recently by the 2500-ft. tunnel, being driven in the property. J. Richard Brown, engineer in charge, reports a sample showing high tungsten content. The ore was opened 200 ft. from the portal.

▶ Contracts have been awarded for grading and foundations for the $9,000,000 aluminum plant which is to be built for the Aluminum Company of America at a point 10 miles north of Spokane. Laurence B. Kuhns, of Pittsburgh, superintendent of construction for the Aluminum Company, states that 20,000 cu.ft. of concrete will be used in the foundation. American Bridge Co. has the contract for furnishing and erecting 6,000 tons of steel which go into the ten acres of building. The plant is designed to produce 60,000,000 lb. of aluminum annually. It is expected 400 men will be employed in operating the plant. The contracts awarded go to Henry Georg and to Clifton & Applegate, of Spokane.

▶ Miners' wages at Grandview and Metaline Mining properties advanced in November to the highest point ever reached in the district and went still higher in December. The advance was 3c. an hour in November and the same in December. The wage is on a sliding scale depending on the price of zinc for the preceding month. The men work six days a week, receiving for the sixth day time and a half for overtime.

▶ E. A. Garber, vice-president and manager of Northwest Magnesite Co., told a mining convention at Chewelah that the magnesite deposits of that district are the largest deposits of that mineral on the Western Hemisphere. He said they are capable of supplying the entire

requirement of steel plants in the United States. These deposits, Mr. Garber said, are entirely crystalline. The colors are black, bluish grey, and red to white, depending on the amount of iron, manganese, and aluminum. The average analysis of crude magnesite is about 3 percent silica, 4 percent calcium carbonate, 1 percent iron oxide, 1 percent alumina, and 91 percent magnesium carbonate.

▶ Fred R. Jameison, a Montana operator, has taken over the Old Dominion, a property in Stevens County which was famous 40 years ago, and believes he can discover new pockets of lead ore which will make this mine a producer once more.

ALASKA

Large Gold Dredge Ready For Next Season

United States Smelting, Refining & Mining Co. obtains modern 9 cu.ft. capacity bucket unit—Investigations for tin in several areas planned for next year

▶ Yuba Manufacturing Co., of San Francisco, has completed a 9 cu.-ft. bucket gold dredge at Nome for the U. S. S. R. & M. Co. The dredge is equipped with many modern features including a heating system and it is reenforced for ice conditions. The hull is 210 ft. long and 72 ft. wide and the distance from the front of the foot bridge to the end of the stacker is 530 ft. It is the deepest dredge in Alaska and will dig 98 ft. below the water. The weight of the dredge is 3,450 tons.

▶ Dr. Phillip S. Smith, chief of the geological service in Alaska, made extensive examinations this season in old gold placer creeks in the Hot Springs area. He collected data on some attractive tin-bearing properties. He says the tin survey in the Territory will continue next year in several areas including Morelock and Grant Creeks, in the lower Yukon and Tanana River localities. Examinations have been under way for some time in the Cape York area, the chief producer of tin in Alaska.

▶ The shipment of tungsten concentrates from the Hyder district of Alaska began early in November from the Riverside mine, operated by J. H. Scott Co. J. C. Roehm, a Territorial mining engineer, reports that several old properties in the district have been found to contain this strategic mineral.

▶ John East, of the United States Bureau of Mines, has 22 men drilling and investigating nickel deposits on Yakobi Island.

▶ Placer mining in many districts has suspended for the winter. The Shonbeck dredge on Ganes Creek was one of the last to quit. It was frozen up Oct. 14.

▶ The University of Alaska is giving soldiers in nearby camps an opportunity to take courses in mining while they are stationed in Alaska. Among the stations which are included are Kodiak, Chilkoot, Fort Raymond, Fort Richardson, Annette Island, and Ladd field.

4 AKINS CLASSIFIERS USED IN NEW TUNGSTEN PLANT BY U. S. VANADIUM CORP.

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The scheelite ores of the Pine Creek property of U. S. Vanadium Corporation contain ALL the common impurities objectionable to tungsten products. The process used to treat these complex ores is the result of four years of research and development.

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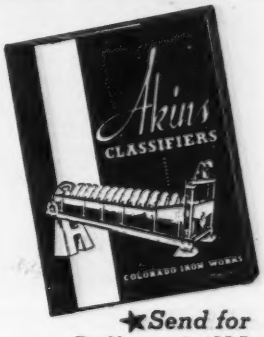

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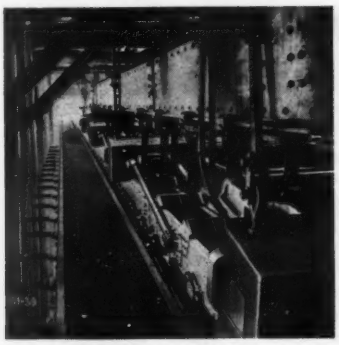

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TRI-STATE

Mine Output of Zinc Slightly Higher

Some producers still shut down because of mines being flooded from heavy rainfall—Kansas Exploration, Inc., suspends operations at Jarrett mines—American Zinc Co. may increase high-grade zinc output from its East St. Louis plant.

▶ Production of zinc concentrates in the Tri-State during the first three weeks of November was recovering gradually from the effects of torrential rains on mining properties throughout the district during October. From the September peak: Aug. 28 to Oct. 2, inclusive—of 9,832 tons weekly average, zinc ore output declined during the next four-week period in October to an average of 8,945 tons weekly. The average for the first three weeks of November was 9,334 tons. At the end of the third week in November the average weekly production for the year to date (47 weeks) was 9,147 tons, which compared with 9,007 tons weekly for the first half of the year. With five weeks remaining, the latter half of the year showed an average of 9,321 tons weekly, and, barring no extreme cold weather, the last six-month period of the year will show a weekly average around this figure.

▶ Among the properties in the district still shut down due to the flooding of underground workings caused by the October rains are those of the Hopi Mining Co. at Thoms Station, old Dividend mine at Spurgeon, Capital Mining Co. at Stotts City, and the Rom Mining Co. near Lincolnville. The High Grade Mining Co. at Chitwood and Fenix & Sons at Oronogo have been handicapped considerably by the heavy increase in the inflow of water underground.

▶ Another factor which influenced the weekly ore output of the district was the closing of the Jarrett mines and mill of the Kansas Explorations, Inc., north of Picher, in Kansas, during the second week of the month. Low-grade ore reserves, together with a price that is not consistent with present-day operating costs, forced the suspension of operations at the properties, according to Ross Blake, of Joplin, Tri-State manager of the company.

▶ Jesse Jones, Federal Loan Administrator, announced on Nov. 17 that Defense Plant Corp., a subsidiary of RFC, at the request of the Office of Production Management, has authorized the execution of a lease agreement with the American Zinc Company of Illinois, East St. Louis, Ill., to provide for facilities at East St. Louis, at a cost of approximately $4,000,000, to be used in the manufacture of slab zinc. The proposed facilities will have an estimated annual capacity of 17,000 tons of electrolytic slab zinc and 7,200 tons of retort slab zinc. Title will remain in Defense Plant Corp.

▶ Pleasant Valley Mining Co. and the Southside Mining Co. are expanding operations at the old Pleasant Valley mines, three miles southwest of Carthage. Associated in the former company are R. S. Hagar and Dr. H. A. LaForce, both of Joplin, and the partners in the latter company are William Laster, Tobe Hunter, and Herbert Cooper, all of Joplin; Ellis Strait, of Four Corners; Albert Danhackle, of Stone's Corner; and John Youngblood, of Carthage. The old mining area, which until this year had been idle many years, has been dewatered and a lower run of high-grade zinc ore is being opened up. Ore is being trucked to the Playter custom mill at Waco for treatment.

▶ Federal M. & S. Co. is sinking a new shaft on a drill hole between the company's Granby-American mill and Crabapple mine, southwest of Granby, according to L. G. Johnson, of Baxter Springs, district superintendent. The shaft will be put down to a depth of about 50 ft., where churn drilling disclosed a run of lead ore. The shaft, which may be sunk deeper later to pick up a run of blende ore, will be known as the Lead shaft.

▶ D. G. Harrison, of Joplin, district manager of the United Zinc Smelting Corp., reports the opening of zinc-lead ore on the 205-ft. level at its No. 12 shaft. The shaft is near the Royal-Jefferson lease boundary line. A 4-in. Pomona turbine pump was installed to dewater the shaft to the 245-ft. level, from where it will be sunk to a depth of 260 ft. to open a lower run of ore.

▶ Ozark Mining Co. has taken a lease on the Henshaw land, a mile south of Baxter Springs, and has reopened a shaft put down a few years ago to the 140-ft. level in the southwest corner of the west 40 of the 80-acre tract. A winze is being sunk in a 90-ft. drift out from the bottom of the shaft. Ore will be trucked to the Eagle-Picher Central mill for treatment. Associated in the company are Louis Myers and John T. Holmes, both of Picher, and Rusty Bullock, of Webb City.

▶ Mining operations have been resumed at the Bingham mine of the Oklahoma Interstate Mining Co., a mile southwest of Baxter Springs, in Oklahoma, according to E. H. Gibb, of Joplin, company manager, who succeeded the late C. F. Dike, of Joplin.

MICHIGAN

Copper Mines Get Back Pay

Employees of some copper-mining companies benefit from higher copper price agreement made with government.

▶ Over $75,000 in back pay has been distributed to more than 1,500 employees of the Copper Range Co., Quincy Mining Co., and Isle Royale Copper Co. The payments were made under agreements negotiated between the companies, employees, and the Procurement Division of the Treasury Department at the time an increase in the price of copper was granted the local companies by the government. The wage agreement provided an increase of $1 per day for mine employees and was made retroactive to

the date the government started taking Lake Superior copper at a higher price than 12c. per pound, the current price. From now on the employees will receive the dollar-a-day increase in the bi-monthly pay checks. Calumet & Hecla Consolidated is not included in this agreement.

▶ The Securities and Exchange Commission on Nov. 21 ordered the listing and registration of the capital stock, $25 par value, of the Mayflower-Old Colony Copper Co. on the Boston Stock Exchange be withdrawn. The registrant failed to file its annual report for the fiscal year ended Dec. 21, 1940. Directors had voted to request the Commission to withdraw its registration "as the company has not the necessary funds to pay for an audit." The corporation was organized under the laws of the State of Michigan and registered under the Securities Act in July, 1935.

IRON COUNTRY

Large Winter Program Gets Under Way

Stripping operations start early in preparation for heavy 1942 mining season—Diamond-drilling program started exploring low-grade iron-ore deposits around Taconite, Minn.

▶ Iron-ore shipments for November amounted to 7,660,987 tons, bringing the total for the season to Dec. 1, to 78,831,787. Shipments for October were 465,063 tons less than for October 1940. This was due partly to the bridge collapsing at the Soo Locks, bad weather on the Lakes, and also the increased movement of coal to the head of the Lakes. Ore movement will undoubtedly extend into December, with the weather being the only retarding influence. Ice breakers are being commissioned to keep the rivers and harbors open and every effort is being made to ship as much iron ore to the furnaces as possible.

▶ All mining companies are making plans for the heavy demand for ore in 1942. The Oliver Iron Mining Co. is planning a heavy stripping program at most of its properties. At the Morris mine, in Hibbing, a screening and crushing plant will be installed. A belt conveyor 1,050 ft. long with a lift of 245 ft. will carry the ore from the pit. The belt is unusual in that it is one continuous belt of steel-cord construction. At the Arcturus mine, in Taconite, Minn., a screening plant and belt conveyor will be installed to expedite mining. A new property to be known as "The Gross-Marble," in the Canisteo district, is to be opened for 1942 production. This ore will be treated at the Trout Lake concentrating plant at Coleraine, Minn.

Mesabi Range

▶ Approximately 25,000 ft. of test drillings will be done throughout the taconite area, with four crews in the

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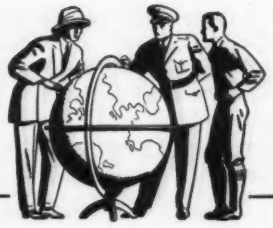

Helping to Speed Defense on 4 Continents

Within the past few weeks a large shipment of Pan-American Jigs left for Malaya to be used in tin recovery. A substantial order from the Belgian Congo was recently filled for the same purpose. In Bolivia, Canada, the United States and several other American republics, Pan-American Jigs and Flotation Machines are speeding defense efforts by getting out strategic minerals faster and more economically.

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▶IRON COUNTRY

field. the drilling and exploration work will furnish the basis for a further extensive program of research and experimentation involving mining interests and the University of Minnesota mines experiment station in an attempt to develop the cheapest and most practicable method for treating this low-grade ore to make it commercially useful. Present reserves of high-grade ore in the Minnesota iron ranges are estimated to be adequate at the present rate of consumption for not more than 20 years. Low-grade deposits are known to exist at least fifty times as great as those of high-grade ore, sufficient for many years under current production schedules.

Action by the Minnesota Legislature during its last session also has aided present plans to develop taconite. Under previous valuation laws, had the mines appraised taconite as of commercial value they would have been taxed on the basis of existing deposits. Now land containing low-grade ore, under a law passed with the support of the range legislators, is taxed at $1 an acre until mined, after which the levy is made on a tonnage basis on the concentrate which is made from it.

▶ Butler Brothers have commenced stripping a new property in Taconite, Minn. It will be known as the "Barbara Mine" and was formerly called the Adgate. A washing plant and belt conveyor system are to be constructed and the mine is expected to be in production in 1942.

▶ M. A. Hanna Co. is starting development of the Mississippi No. 1 Reserve at Keewatin, Minn. This property will be stripped and operated as an open-pit mine with the ore being treated at the Mesaba-Chief concentrator.

▶ Inter-State Iron Co. is carrying on a heavy stripping program at the Hill-Annex mine, removing about 400,000 cu. yd. of material per month.

▶ Cleveland-Cliffs Iron Co. is carrying on a stripping program at the Holman-Cliffs mine, in Taconite, and also at the Canisteo mine, in Coleraine, Minn.

▶ Plans for bringing Minnesota's vast deposits of low-grade iron ore into the National Defense and industrial picture moved forward Nov. 22 when E. J. Longyear & Co., exploration engineers and contractors of Minneapolis, began drilling test holes in the taconite area near Mesaba, Minn. Drilling operations are being conducted for the Erie Mining Co., subsidiary of the Bethlehem Steel Co., and the Youngstown Sheet and Tube Co.

Marquette Range

▶ Inland Steel Co. has suspended operations for about a month at its Greenwood mine for the purpose of erecting a new steel headframe. The Lakeside Bridge & Steel Co., of Milwaukee, has the contract for the work. A new screening unit will also be installed near the headframe.

▶ Shipments from the stockpiles of hard hematite ore at the Champion mine, at Beacon, have stopped for this year. The property is owned by Oliver Iron Mining Co.

▶ A 2,100-cu.ft. Ingersoll-Rand compressor was recently installed at the Mather mine of the Cleveland-Cliffs Iron Co. The cage-hoist, manufactured by the Nordberg Co., is now being installed and the hoist for the large skip is expected to be ready for shipment to the mine in February or March.

▶ A hoist and compressor are being shipped from the Berkshire mine, at Iron River, to the Ironton and Puritan mines in preparation for a diamond-drill program to determine ore extensions at these properties below their deepest workings. W. A. Knoll, 709 E. Ayer St., Ironwood, Mich., is general superintendent.

Menominee Range

▶ Sinking the new shaft at the Sherwood mine of the Inland Steel Co. is expected to be completed in December. Bottom will be at 1,200 ft. Work has started in cutting the 1,200-ft. station. R. D. Satterly, of Iron River, Mich., is Menominee Range superintendent for the Inland Steel Co. and Victor Backels is in charge of operations at the Sherwood mine.

▶ The three Davidson mines of the Pittsburgh Coke & Iron Co., in Mineral Hills, at Iron River, Mich., have increased operations from a five- to six-day week.

MEXICO

U. S. Agrees to Buy Mexican Silver

Economic relations between the United States and Mexico are expected to improve following arrangements to purchase about 6,000,000 oz. silver per month at 35c. per oz.

▶ A six-point agreement has been reached between the United States and Mexico, aimed at strengthening economic relations, it was announced officially in Washington on Nov. 20. The agreement covers property claims, petroleum, trade, stabilization of the peso, credits, and silver. In reference to silver, the U. S. Treasury Department has agreed to purchase Mexican newly mined silver directly from the Government. Settlement for silver acquired will be made on the basis of 35c. an ounce, under the Silver Purchase Act of 1934. The Treasury is expected to buy about 6,000,000 oz. a month, based on the agreement just concluded.

▶ Some optimism was noted in the Mexican mining industry as winter approached, with prospects of national benefits that are conceded to accrue from the loan the Export and Import Bank is arranging for this country and the announcement by the National Railways that the Mexican Government, its administrator, is to spend 200,000,000 pesos ($42,100,000) next year in a general renovation of this railroad, Mexico's greatest railway, as it serves three fourths of the republic.

▶ President Manuel Avila Camacho, who acted as umpire in the case, settled a conflict between two factions of the cooperative miners who for some time have had charge of properties in the Ojuela district, Durango, as to which should be officially recognized. The President ordered that the larger group must have such recognition. Work resumed with this ruling.

▶ Several mining companies in the Pachuca, Hidalgo, zone, including Real del Monte, were spared embarrassment that threatened their suspension when action of the Federal Government averted a strike against the Pachuca electric power company that had been called for Nov. 15.

▶ Mexico's largest private bank, the Banco Nacional de Mexico, S. A., backed by French capital, asserted in a pessimistic review of the industry that discipline and good will of labor is one of the things the business must have if it is to improve.

▶ Sinaloa Mining has found so successful the small cyanide plant it has been operating as an experiment on its properties at Cosala, Sinaloa, that it has decided to retain the service. F. F. Gerard, of Bellingham, Washington, and Ralph B. Le Cocq, of Lynden, Washington, are president and secretary-treasurer, respectively of this company, which is officially known in Mexico as the Cia. Minera Sinaloa, S. A. William T. Dumbleton is the resident consulting engineer.

▶ Work in the Dos Carlos mine, Pachuca, Hidalgo, has been hampered by the explosion of an air compressor that injured the operator and is reported to have resulted from cheap oil used in the compressor department. This mine is owned by the cooperative society that for nearly four years has been working properties placed in its charge by the Cia. Minera Dos Carlos, S. A., a British firm, which found operations so unprofitable that it had to quit.

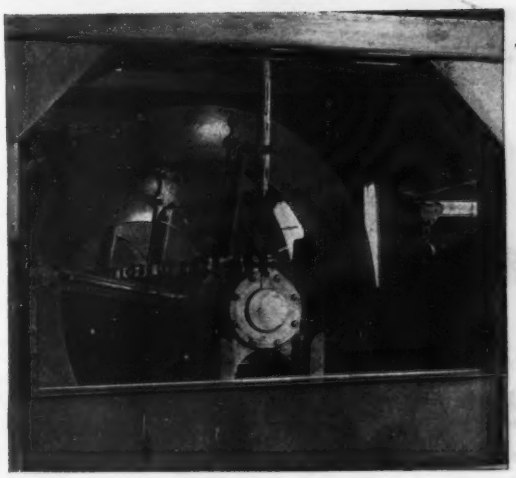
▶ Granting a general daily pay raise of from 10 to 15 centavos (2.25 to 3.40 cents) and agreeing to donate 6,000 pesos ($1,035) and provide 3,600 pesos a year for medical services, settled a work conflict for the Cia. Minera del Boleo, S. A., Santa Rosalia, Baja California, a French copper-mining enterprise, of which August Nopper is managing director. The cash gift is to enable the workers to found and run a consumption cooperative society.

▶ As experts have reported that the proposition of installing modern and more costly equipment to supplant that which forced the plant to suspend work some time ago is feasible, operations are soon to resume at El Salto iron-steel foundry, El Salto, State of Mexico, according to the Ministry of National Economy.

▶ The highest mining accident death indemnification to be paid in Mexico recently, that of 14,000 pesos ($2,950), was settled by the Asarco company with the widow of Antonio Jaquez, a worker killed in one of its Chihuahua mines.

▶ Charged with being the moving spirits in the sensational plot to smuggle a large amount of mercury, antimony, and other metals and minerals that are con-

UPPER BEARING



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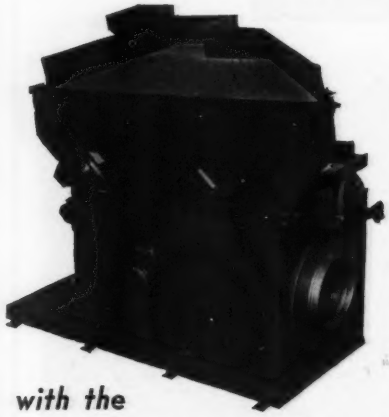
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▶MEXICO

traband under the economic defense agreement between Mexico and the United States, to Japan on a Nipponese freighter from Manzanillo, chief port of Colima, A. Z. Phillips, of Rosendo Soma, and Luis Basurto and Antonio Llorens have been committed for trial at the port by the Federal Attorney General. The trial is to open in December.

▶ Production was at a generally higher level during the first seven months of this year than it was during the like period of 1940, though there were decreases in the output of gold, lead, and some of the minors in the 1941 term. It is shown by the following figures of the Ministry of National Economy, amounts in kilograms:

Product	1940	1941
Gold	15,826	15,644
Silver	1,485,759	1,511,360
Copper	22,296,587	30,280,117
Lead	120,867,200	98,450,313
Zinc	64,248,763	69,181,347
Antimony	7,675,996	7,119,682
Graphite	7,209,868	10,507,440
Mercury	185,265	593,343
Arsenic	6,879,933	7,409,891
Cadmium	524,984	403,467
Tin	214,933	132,236
Molybdenum	397,069	561,606
Iron	40,252,072	39,608,508
Vanadium	14,760	
Manganese	195,474	233,089
Bismuth	83,246	63,826
Tungsten	52,774	56,297
Coal	524,377,169	479,047,087

▶ Emergency crews released the 27 miners who were trapped by a cave-in of undetermined origin on levels 350 and 375 of the San Rafael mine, Pachuca, Hidalgo. The men were released after they had been imprisoned for nearly a day. The Pachuca authorities are investigating the accident. Work was quickly resumed in the mine.

CANADA

Miners Returning To Work at Gold Mines

Strike called as conference between union officials and mine operators fails to find a solution to difficulties wherein CIO seeks union recognition and to be sole bargaining agent for miners in eight gold mines in Kirkland Lake district

▶ Labor trouble in Kirkland Lake came to a head on Nov. 18 after four months of impasse. Miners walked out on strike at eight of the largest gold producers, following a vote in which 37 percent of the men were reported opposed to the strike. A last-minute conference at Ottawa between the operators and representatives of the Mine, Mill and Smelter Workers Union No. 240 failed to work out a solution. Two of the mines were closed down and reduced production was carried on at Wright-Hargreaves, Teck Hughes, Sylvanite, Toburn, Macassa, and Bidgood. Picketing was no longer peaceful by the end of the first week and, in response to a call for assistance from R. J. Carter, the reeve of Kirkland Lake, to supplement the 14 local constables on guard at the mine gates, a special train was sent up from Toronto with a large number of Provincial police, increasing the patrol to more than 200 men. The only issue at stake is union recognition, as the sole bargaining agency of mine workers and the CIO action here in testing the results of a strike is regarded as making Kirkland Lake a straw in the wind to guide a wider labor program in other Canadian mining camps. Ten men and a woman had been arrested on charges of intimidation after the first ten days of the strike. Operators reported that the number of men at work had increased to 970, and on the tenth day the combined output of all eight mines was 1,760 tons. As tension increased, Premier Mitchell Hepburn promised to keep things under control "at all costs . . . if we have to recruit an army."

▶ The Wartime Cost-of-Living Bonus to be paid by the mines was increased to $3.68 a week to all workers earning $25 weekly, after Nov. 15. The index of the Dominion Bureau of Statistics shows an increase of 14.6 percent in general living costs since August, 1939. The increase for September amounted to 114.7 to 115.5.

▶ September gold production for the Dominion reached a total value of $17,535,773, compared with $16,984,082 in the same month a year ago. A decline of almost $450,000 from the previous month may be evidence of a gradual falling off from the peak reached this year. For the first nine months of 1941 Canadian gold output has totaled 4,005,205 oz., valued at $154,600,393, compared with 3,941,401 oz., valued at $151,743,938, in the same period of 1940.

▶ Special depreciation and tax credit allowances under the Income War Tax Act and the Excess Profits Tax Act are to be granted International Nickel, Falconbridge Nickel, and Normetal. Enlargement of these three plants on a wartime basis will provide for assistance to Falconbridge through a write-off allowance of a cent a pound for all metal sold, to take care of an estimated capital outlay of $901,920. Normetal will be able to write off capital expenditure of $165,000 in two years. International Nickel is planning to spend $29,000,000 for expansion in the next two years, followed by an expenditure of $3,000,000 in 1944–45. This program will add about 50,000,000 lb. of nickel annually to the 1940 rate of production. Increased exports that will result are expected to earn at least $15,000,000 for the credit of the Foreign Exchange Control Board, by the end of 1945. In the next five years, International Nickel will be able to amortize all but $7,000,000 of the total expenditures for plant expansion.

▶ Further development of the important northeast ore zone at McKenzie Red Lake Gold Mines is to be carried out before the mill capacity is increased, according to a report by the president, W. G. Armstrong. This year the mill has been averaging 240 tons a day, compared with 209 tons in 1940. New ore in the northeast zone is not sufficiently opened up to indicate how a mill increase would result in the most efficient operations.

▶ Chesterville Larder Lake Gold Mining Co. has completed shaft sinking at a depth of 1,250 ft., with five new levels

ready for development. The ninth level station has been cut at 1,175 ft. Excellent ore has been indicated at the lower horizons by diamond drilling, both on the 1,300-ft. level in workings on the adjoining Kerr-Addison property, and on the first two of the new levels, where crosscuts are approaching the "C" orebody. L. T. Postle is mine manager.

▶ Tombill Gold Mines, in the Little Long Lac area, has been worked for the past year as a salvage operation after exhausting the possibilities for discovering new ore. However, since the July estimate of 44,000 tons of ore reserves, sufficient tonnage has been added by extending old stopes to assure production for at least another six months. The deeper ore is being mined and milled by Bankfield, paying Tombill a royalty of 50¢ a ton on ore grading up to $9, and dividing any higher values equally between the two companies. Capacity of the Tombill mill is 130 tons daily. Charles Caldwell is mine superintendent.

QUEBEC

▶ Joseph R. Norrie, manager of East Malartic Mines, reports that ore of better than mine average has been located in drifting west toward the Sladen Malartic boundary, on the deepest of the block of new levels at 1,570 ft. The new ore underlies the Sladen fold within 150 ft. to the west of the shaft, and has been opened for a continuous length of 100 ft. The grade is between $8 and $10 per ton.

▶ Joliet-Quebec Mines, formerly the Brownlee property, adjoining Noranda on the west and north, is considering an additional program of diamond drilling as recommended by E. K. Fockler, consulting engineer. In the spring of 1940 a deposit of low-grade copper was discovered, estimated at 1,000,000 tons of highly silicious replacement material averaging about 1 percent copper and 0.01 oz. gold. It is believed that 75 percent of the material would be commercial grade ore with copper at 10¢ a pound, allowing for the desirable fluxing properties allowed by a 73 percent content of free silica.

▶ In October new records were set by Sigma Mines in both the tonnage milled and gold output. Averaging 1,100 tons daily, the mill treated 34,122 tons of ore to produce a total of $262,152, an average recovery of $7.68 per ton. J. H. Stovel is general manager and J. B. Redpath is mine superintendent.

▶ Early in November Waite Amulet Mines went into regular production at the new rate of 1,500 tons a day, including 300 tons of zinc ore to produce concentrates for the American market. This is strictly a war measure that is not expected to provide any profits. Amulet's first dividend was paid in July, 1940, and since then has disbursed a total of 50¢ a share. The company is controlled by Noranda.

▶ A report on Lake Dufault Mines by J. G. MacGregor, president and consulting engineer, shows diamond drilling during the past six months has found conditions of structure and mineraliza-

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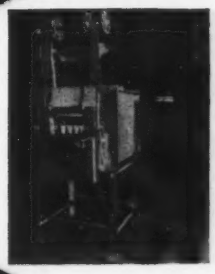

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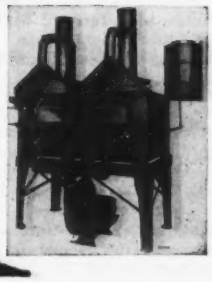

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Beattie Gold Mines, Ltd., 30 miles north of Noranda, in Duparquet township, Quebec, is treating around 2,000 tons of gold ore per day in its cyanide plant at the right. Roaster plant and arsenic storage tanks are in the background

▶CANADA

tion similar to the Lower "A" orebody on the Amulet-Dufault property. Drilling from the 1,100-ft. level, two out of three 1,000-ft. holes cut heavy pyrite. Lake Dufault has received $70,000 in dividends from its share of the Lower "A" orebody and will use part of its treasury to continue exploratory drilling.

BRITISH COLUMBIA

▶ Homeward Mines, Ltd., newest Zeballos producer, which had been milling on a 24-ton per day basis, increased to 32 tons per day in November. The rated capacity of 50 tons per day is expected to be reached soon. The proposed increase is, however, dependent on the management being able to secure and retain a larger crew than is at present employed. Average mill-head assays (of sorted ore entering the plant) were $20 per ton during October; mill-tailing averaged only 0.01 oz. gold per ton.

▶ Nearly $60,000 has been subscribed by Pioneer Gold Mines of B. C., Ltd., and Anglo-Huronian, Ltd., to bring Musketeer Mines, Ltd., Bedwell River property, to the production stage. Musketeer shareholders have authorized a loan of $100,000 for this purpose.

▶ Howe Sound Co., which operates the Britannia Mining & Smelting Co. in British Columbia, in addition to activities in the State of Washington and in Mexico, had net income of $589,150 for the quarter ending Sept. 30, bringing the total for the year to date to $1,282,178. Gross value of metals sold during the quarter was $4,729,269; during the nine-month period, $11,755,118; net value of metals sold, less contracted marketing, smelting, and other costs, was $2,691,973 for the quarter, and $7,221,292 for the three quarters; operating profit, after deducting costs, was $973,237 for the quarter, and $2,217,389 for the three quarters; total net income, plus miscellaneous income, was $981,474 for the quarter, and $2,244,689 for the nine-month period. Income per share, less provision for taxes, was $1.23 for the quarter and $2.69 for the year to date. A dividend of 75¢ per share was paid on Sept. 30.

▶ Lake Logging Co., Ltd., has acquired the plant and property of Fraser River Golds, Ltd., on the Fraser River, near Hope, B. C., and plans to conduct dredging operations.

▶ Hedley Mascot Gold Mines, Ltd., Osoyoos Mining Division, milled 17,841 tons of ore in the quarter ended Sept. 30. Net income, after provision for taxes, development, and exploration, was $56,948; net earnings per share, before depreciation and depletion, were 2.51¢; and average heads for the quarter were 0.36 oz. gold per ton. Dr. Victor Dolmage, consulting geologist, in a quarterly report to shareholders, states: "Since the last quarterly report, development work at Mascot has consisted in the exploration and opening up of ore between the 4,500 and 4,800 levels in the general vicinity of the 4,510 raise and in extending the surface explorations on the Galena, Copper Chief, and other adjoining claims. The results obtained in the former locality have been satisfactory and in some respects have exceeded expectations. Particularly encouraging is the depth to which ore is being found below the hot sill in this locality. The surface explorations have indicated some favorable areas, which, however, are too deep to be tested from the surface and must therefore be explored by a new low-level tunnel. Investigations are now being carried out for the purpose of assisting in the selection of the most suitable locality for such a tunnel.

▶ A final-treatment plant has been completed at Silver Creek siding, in the Lardeau district, to treat the concentrate recovered from the Regal Silver property. This plant produces a refined concentrate of from 60 to 70 percent WO_3 which is shipped to the Atlas Steel Co., Welland, Ontario.

▶ New York interests, headed by Col. J. A. Maller, have taken an option on the Lightning Creek property of Consolidated Gold Alluvials, Ltd., at Wingdam, in the Cariboo district. Under the terms, the optionee is to receive 70 percent of all net profits resulting from

operation. Advances totaling $1,000,000, made by W. J. Yapp, of London, England, prior to the cessation of the last operation, represent a first obligation on the part of the company, and must be defrayed before any of the optionee's 30 percent of the profits is distributed to shareholders. First work is expected to include Keystone drilling, to test the ground. It is understood the new operators are prepared to expend as much as $250,000.

▶ Shareholders of Relief-Arlington Mines, Ltd., on Nov. 15 approved a resolution, calling for voluntary liquidation, and appointed D. G. Marshall as liquidator.

▶ Bayonne Consolidated Mines, Ltd., has completed two diamond-drill holes, testing the possibilities below the 8th or bottom level. Values of 1.24 oz. gold per ton have been found over width of 1.4 ft. at a depth of 100 ft. below the 8th level. Recently, the extension of the 8th level drift into the mountain had developed one extensive and several smaller orebodies. The present discovery lends credence to the belief of the continuance of ore at depth.

SOUTH AMERICA

Surinam (Dutch Guiana) Increasing Bauxite Output

Important aluminum ore deposits are being protected by American military units—Molybdenum production in Peru no longer going to Japan

▶ In a recent issue of the Foreign Commerce Weekly, the Department of Commerce describes the bauxite mining industry in Surinam—more commonly known as Dutch Guiana. Valuable deposits of bauxite, the only commercial source of the metal aluminum, were discovered in Surinam 25 years ago in the Moengo region, where an American plant is located and from which the bulk of the output of bauxite in Surinam has come. Today bauxite is the principal product of the country and in point of volume and value represents nearly 90 percent of the country's total exports. Surinam, together with British Guiana, is important as the almost inexhaustible source of bauxite of unsurpassed quality, and these resources of South America assure the Western Hemisphere independence with respect to the raw material for making aluminum. All of Surinam's bauxite is now shipped to the United States, increasing from 125,000 long tons in 1932 to 586,000 tons in 1940. (The United States supplies only one-third to two-fifths of its domestic requirements from bauxite produced in the United States; 95 to 100 percent of all United States imports of bauxite come from Surinam.) Exports of bauxite from Surinam in specified years from 1929 to 1940 were as follows:

	Long Tons		Long Tons
1929	207,000	1939	477,000
1932	125,000	1940	586,000
1938	387,000		

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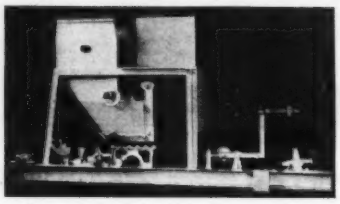

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WRITE FOR CATALOGUE #7

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Wide World Photos

Modern American mining equipment is employed on the bauxite properties of the Surinam Bauxite Mining Co. in Dutch Guiana, South America. The ore found in these mines is of unsurpassed quality, and about 60 percent of the aluminum requirements in the United States are supplied from ore deposits in Surinam

▶SOUTH AMERICA

bauxite there is the result of the flow from new deposits just developed and it is expected that these new resources may contribute as much as one-half the total of anticipated imports to the United States from that country, or about 1,200,000 long tons in 1941. In the Moengo region, 104 miles up the Cottica River, is located the original plant of the American bauxite interests, from which the bulk of the bauxite exported from Surinam has come to date. This plant is undergoing expansion, and at the new port of Paranam another plant has been constructed to handle ore from the Para River and Para Creek district. Modern methods are employed in the exploitation of the deposits. Transportation to the United States, the Gulf ports, and Canada is by water. Practically all of the bauxite industry in Surinam represents the investment of American capital and the promotion of American interests. In 1941 a large Netherland company which was originally engaged in mining bauxite in the Netherlands Indies also began construction of a plant on the river near Paranam, and this plant will be used for treating bauxite from the Para Creek district. A strong American military unit is now being established in Surinam to protect these deposits. This action was taken following agreement of the Netherland Government at London and the Brazilian Government and the United States that this step be taken.

ARGENTINA

▶Announcement was made in Buenos Aires on Nov. 27 that the United States will undertake to buy all tungsten produced in Argentina during the next three years up to a maximum of 3,000 tons tungsten oxide per year. The Metals Reserve Co. will pay a basic price of $21 per short ton unit of WO_3, with deductions or premiums for lower or higher grades. Output of tungsten from Argentina is estimated around 2,000 tons per year, and with this promised market for the metal producers are expected to increase output. The principal producer of tungsten is Sociedad Minera Argentina, S. A., whose largest mine, the Condores, is at Concaran via Villa Mercedes, Province of San Luis.

BRAZIL

▶Andriano Marchini, president of the Technological Research Institute of Sao Paulo, has recommended an immediate expansion of the Apiai lead works from a capacity of 10 tons per day to 60 tons per day, and has received State credit for the program. This project is part of the growing metallurgical industry of Brazil, the other developments being the steel plant of Volta Retonda and the aluminum works of Ouro Preto.

The lead works of Apiai, erected last year under the supervision of the Technological Institute, is a government-owned plant to treat lead minerals produced by numerous exploration enterprises within a radius of 35 km. Prior to this undertaking Brazil had only two medium-sized lead works in Parana, in the southern part of the country—Plumbum, S. A., with a capacity of 2,000 tons per year, and Sociedad Mineracao das Furnas, 4,000 tons per year. Both previously exported their production to the United States but will now ship to Apiai.

The government of Sao Paulo has asked the federal authorities at Rio de Janeiro to take steps to obtain from the United States supplies and equip-

ment for the construction of a zinc plant in Apiai. A credit of $1,500,000 has been allowed for the construction of a zinc plant in Apiai. A credit of $1,500,000 has also been allowed to the Technological Research Institute, which is to supervise all government-owned key plants for the development of the mineral resources of the country. This policy arises from the scarcity of private capital, and the desire of the State to assist numerous small producers of metals.

VENEZUELA

▶ Asbestos mining concessions will be exploited in Tinaquillo, State of Cojedes, if the orders for equipment placed in the United States can be obtained. Production of asbestos is planned at the rate of 10 metric tons daily.

PERU

▶ Production of molybdenum in Peru is virtually controlled by one company called Peru Molibdeno, with properties in Juaja, Department of Junin. Concentrates shipped in the first six months of 1941 totaled 206.3 tons, averaging about 80 percent MoS_2, according to the U. S. Department of Commerce. A contract held by the company with Japanese interests for the entire output expires in December, 1941. The entire output in 1942 has been contracted for by the British Government.

CHILE

▶ According to a Department of Commerce report, shipments in August, 1941, through the Chilean port of Antofagasta, of Bolivian tin and tungsten ores destined for the Metals Reserves Co., in Washington, D. C., total 4,658,860 kg. and 289,297 kg. respectively.

SPAIN

Mineral Deposits Being Appraised for Defense

The Spanish Government has established an Advisory Council on Minerals of Military Significance, that is charged with the study of mineral resources that might be of military interest and directed to make recommendations for their development. The order covering this move was published in Spain, Sept. 17. At the recommendation of the Council, the development of all deposits of tin, copper, aluminum, zinc, manganese, tungsten, molybdenum, nickel, chromite, vanadium, titanium ores, and zirconium has been declared of "interest for national defense." The council has the power to exercise direct or indirect control over mines producing such minerals. The Minister of Labor and Director-general of Labor have already paid the British-controlled copper mine at Rio Tinto a visit.

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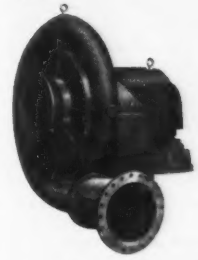

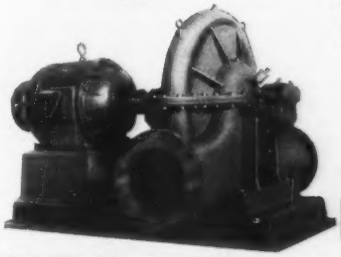

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AFRICA

Diamond Concessions Being Surrendered

DeBeers Consolidated Mines unable to meet requirements of continued prospecting in Southern Rhodesia—Prospects believed good for a gold field in Orange Free State

▶ It is announced that DeBeers Consolidated Mines is surrendering its diamond concession in Southern Rhodesia to the government. The option was acquired from the British South Africa Co. in 1892. After the latter sold its mineral rights to the Southern Rhodesia Government the concession was continued. In 1936 DeBeers agreed to perform a specified amount of prospecting annually for 20 years, surrendering a portion of its concession each year. The war has made such prospecting difficult, and the people of Southern Rhodesia are eager for the State to own all mineral rights in the Colony. When the government has passed the necessary legislation the surrender will be completed.

▶ Louis Marks, chairman of African & European Investment Co., stated at the 36th annual meeting held recently that drilling in the area adjacent to Western Holdings, and taking a general view, encouraged the belief that an important gold field exists in the Orange Free State. His company has put down six boreholes, three of which revealed payable values, two others notable gold values, and one, because of bad ground, which did not cut the reef. The holes drilled stretch across an area of four miles. The reef was cut at depths between 2,000 and 4,000 ft.

▶ Witwatersrand's gold production showed a slight reduction in August. The chief decrease was over 20,000 oz. in the output of Simmer & Jack, which had a breakdown in the hoist of one shaft and a winding accident at another.

▶ Comparatively little is yet known of the characteristics of the new reef found in the Western Rand areas, known as the Carbon Leader, whether it should be expected to carry values in more or less regularly spaced shoots or in irregular patches. As a result of its being found so sensationally rich in the Blyvooruitzich workings, the advisability of drifting on it in those mines where it was found lean is being considered. It is rumored in Johannesburg that a total of 1,500 ft. of drifts in Blyvooruitzich show an average of 1½ oz. of gold per ton. The sources from which these rumors come is causing them to be taken seriously, as they are by no means extravagant. It will be remembered that the official report stated that in the June quarter 205 ft. opened on the 5th level averaged 4 oz. and 2½ dwt. per ton.

▶ H. R. Hill, engineering consultant to the Union Corporation, has prepared an estimate of the gold production of South Africa during the coming 30 years. He expects the high point to be reached in 1942, with an output of 15,045,000 oz., then a very gradual decline to 14,360,000 oz. in 1950. After that it will shrink more rapidly, coming down to 7,384,000 oz. in 1970, just over half, 3,334,000 oz., being produced by new mines.

▶ Ashanti Goldfields, the leading producer of the Gold Coast mines, having an output of some 265,000 oz. of gold annually, announces that it will reduce its mine output by 1,000 tons monthly. The object is to prevent a further reduction of its stock of explosives. Regular shipments can no longer be depended on. This is due to the irregularity of the movements of cargo ships. The managements plans are to crush about 18,000 tons and produce about 20,500 oz. monthly. September output was 22,003 oz. Ashanti's dividends came to 80 percent last year against 95 percent the year before. This year two interims of 22½ percent each have been paid. No other mining company has so far reported a threatening shortage of explosives.

▶ Unofficial reports coming from Africa state that the Northern Rhodesia copper mines are establishing new high records of production. It is said, also, that they are delivering their copper in shapes which save time and labor for the munitions manufacturers—evidently no longer merely in ingots, cakes, and bars. Every effort is being made to supply the war demand.

AUSTRALIA

Mt. Morgan Stripping Operations Retarded

Heavy rainfall during most of year slows progress—Gold output in Western Australia for first eight months this year behind figure of same period in 1940

▶ Mt. Morgan, Ltd., has had a difficult year, but, despite adverse conditions, earned a working surplus of £A157,667. The main feature in gold-mining operations at present is the removal of overburden to make the ore reserves of 7,000,000 tons available for open-cut mining. The ultimate depth to which this work will be carried will be 800 ft. from surface. During the year 1,364,887 tons of overburden was removed and dumped, the ratio being 1.58 tons of overburden to 1 ton of sulphide ore milled. To maintain the required quota of reserve ore it is essential that the ratio be 2 to 1 for the next twelve months. Much of the overburden carries payable gold values and is sent to the mill, the greater part of the ore treated during the year being auriferous overburden. Working conditions were rendered difficult by an abnormally wet season, extending over seven months, which seriously interfered with operations in the open-cut. In addition, much of the material treated had been affected by the old mine fire of some twenty years ago, and was extremely acid. Mining under wet weather conditions was very troublesome and treatment was difficult, as much as 50 lb. of lime per ton being required, the average consumption being 24.82 lb., against 16.76 lb. in the previous year. Mill maintenance

was also increased considerably, but this has been overcome by gradual replacement with acid-resisting equipment. Despite difficulties in operation and the increased cost of stores and wages, as well as the more serious conditions due to war, costs were substantially reduced. Ore and overburden are handled on the open-cut benches by mechanical shovels, the ore being dropped down passes to the crushing station at the 650-ft. level, from which it is hauled to surface through the main underlay shaft. Electric power is generated by steam-driven units, fuel being obtained from the company's coal mines in Dawson Valley.

NEW SOUTH WALES

▶ Small-scale gold operations at Lucknow, in the Orange district, west of Sydney, have been carried on intermittently for several years. The district first became prominent through the success of the Wentworth Proprietary Gold Mines, which, to the time of closing down in 1900, produced approximately 200,000 oz. of gold. Rich gold deposits were invariable associated with calcite and were found at and near the junction of the main and branch fissures, especially adjacent to the serpentine. The latest development on the field is reported from the Lucknow Central Gold Mine, where oxidized ore has been met on the contact, values being reported as 200 oz. of gold per ton. Further development is essential to establish the actual importance of the find.

FIJI ISLANDS

▶ Emperor Gold Mining Co., and Loloma (Fiji) Gold Mines announce that they are acquiring between them a controlling interest in a company being formed in Fiji to carry out gold prospecting and exploration over a large concession area on the Island of Guadalcanal, in the British Solomon Island Protectorate.

All the shares of Emperor Gold Mining Co., which was registered in Fiji, are owned by Emperor Mines, of Melbourne.

NEW ZEALAND

Deeper Shaft Planned by Blackwater Mines, Ltd.

Company is now recovering arsenic from roasting furnace fumes—Some gold-dredging companies report

▶ Blackwater Mines, Ltd., at Reefton, treated 49,020 tons of ore during the year ended Dec., 1940 for a yield of 22,836 oz. of gold. Ore reserves were estimated at 100,077 tons having an average grade of 9.87 dwt. per ton. The results of development work were satisfactory, the most important feature being the lengthening of the reef at and below No. 11 level. It is now intended to sink the main shaft 125 ft. to No. 15 level. The new mill is reported to be running well and the equipment has been increased by the construction of a

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▶NEW ZEALAND

settling chamber for the recovery of arsenic from the roasting furnace fumes. The new mill has a capacity of 250 tons of ore per 24 hr. Ore is ground in a 6 x 6 ft. Marcy mill to which cresylic acid, pine oil, and xanthate are added for flotation. From the mill, the product is pumped over primary strakes, the concentrate being treated in an amalgamating barrel. The strake discharge passes to a rake classifier and the oversize to an 11 x 5 ft. ball mill grinding to 60 mesh, the product being straked. Overflow from the second series of strakes passes through a surge tank to six Fagergren flotation cells. Concentrates are thickened, filtered, dried, and roasted in an Edwards furnace. The calcine is cooled, agitated, thickened, and filtered on an Oliver filter. Merrill-Crowe precipitation is used.

▶ Much of New Zealand's gold production comes from the operations of large bucket dredges. Two of the principal companies have announced the results of their work for the year ended March 31, 1941. Barrytown Gold Dredging, Ltd., earned a profit of £A18,345 for the period and paid a first dividend of 6d. per share.

The bucket ladder is 110 ft. long and is equipped with 12-cu.ft. manganese steel buckets. Black sand is recovered in 18 Bendelari jigs in two banks of nine. The concentrate is passed over plush-lined tables and the enriched concentrate is barrel-amalgamated. The total power installed is 820 h.p.

▶ Arahura Gold Dredging, Ltd., operates an electric-powered dredge equipped with 18-cu.ft. buckets and having a designed capacity of 400,000 cu.yd. per month. In the year reported a profit of £A64,821 was earned and three dividends, each of 1s. per share, were paid. The average gold recovery per cubic yard since operations commenced to March 31, 1941, was 1.77 gr. as compared with the average bore value of 3.8 gr. for the same area, the yardage treated being 6,813,000. Operating conditions were generally normal and ground was easily worked.

PHILIPPINES

New Chromite Deposits Being Developed

Two groups of claims on island of Mindanao to be exploited—Mindanao Mother Lode mine, in Surigao, milling richest ore in the Philippines

▶ A new chromite field is being developed in Oriental Misamis, on the west central side of Mindanao. Though existence of chromite there has been known for years, little development work was done, on account of inaccessibility. With insistent demand from the United States for chromite for defense needs exploratory work has been pushed. Already some 14,000 tons of high-grade ore averaging over 50 percent Cr_2O_3 have been mined and stockpiled from one group of claims. A second group 20 km. distant has been acquired by a strong operating company which is rushing construction of a road, about 17 km. in length, to the property. Production is expected early next year, and chance for developing new lenses appear to be good.

▶ The Surigao mining area, in northern Mindanao, has received much attention on account of the sensational increase in the production of the Mindanao Mother Lode Mining Co. to a position where it now ranks second in gold output among Philippine gold mines. Beginning in May the mill heads at this mine increased in value from about 35 pesos per ton to over 100 pesos. In September, the mill treated 8,600 tons for a gold production of $1,131,180, with average recovery of 131.53 pesos per ton. This per-ton value is probably unexcelled by any gold mine in the world that is milling a comparable tonnage of ore. The mill makes a flotation product with a high concentration ratio, that is shipped to the States for smelting. Despite the richness of the ore, no free gold is encountered that is visible to the eye. Although veins are generally narrow, requiring careful stoping to prevent dilution, costs are maintained at a creditable figure. Production comes in the main from the Tabon-Tabon vein, which has been developed to the 700 level, with values holding up extremely well. Plans are under consideration to sink to the 1,000 level. A new vein, No. 50, to the east of the Tabon-Tabon, has also furnished some excellent ore.

▶ Busuanga Island, one of the Palawan group, continues to furnish the main supply of Philippine manganese ore for export. Companies operating there include the Luzon Stevedoring Co., Amalgamated Minerals, and Compañia Minera de Filipinos. The latter controls the important Borac mine, currently producing about 1,500 tons a month of high-grade manganese ore. The Amalgamated Minerals Co. has four producers on the island. New manganese discoveries are reported in Camarines Sur that show promise. Milling difficulties have slowed down production on Siquijor Island, formerly site of the largest manganese production in the Philippines.

▶ Lepanto Consolidated shipped 2,195 tons of copper matte on Oct. 21 to an unknown port, the freighter sailing under sealed orders. This was the largest shipment of copper made this year from this mine, which at present is the only copper producer in the Philippines. The shipment was valued at over $900,000. The copper content of the ore milled at Lepanto has been steadily increasing during recent months as well as the gold associated with the copper ore.

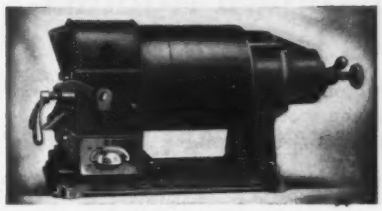
▶ The Federal Loan Agency is reported to be preparing to make loans against stockpiles of iron ore in the Philippines, probably on a limited scale. However, this action will permit a moderate re-

sumption of mining by the four principal iron-mining companies, which have had their market cut off in Japan by imposition of export control measures.

▶ On account of export control measures in force in the Philippines, the Hixbar Gold Mining Co., second most important copper producer in the Islands, has suspended mining operations. With shipments of copper ore to Japan virtually embargoed, there was no other recourse for this company, as Japan has always been the only buyer for the grade of ore produced at the mine. The mine has no concentrating plant, and the ore as mined, carrying about 10 per cent copper with fair gold values, has been shipped direct to Japanese smelters. According to an official statement by the company there is small possibility of resumption of shipments to Japan until the war is over. Present ore reserves are estimated at 18,152 tons averaging 8.26 percent copper. During the last six months of operation ended July 31, the company realized a net profit of $256,199 from a total operating income of $1,020,640. The company has been a consistent dividend payer.

▶ Strong efforts are being made to develop a source of gypsum rock in the Philippines for the use of the local cement plants. Despite apparently favorable geological conditions, no gypsum deposits of commercial size have yet been discovered in the Islands. Gypsum occurs frequently with the gold ores in the Baguio field as a vein filling and some small rather impure lenses have been found in other parts of the Philippines, but they do not warrant much capital expenditure. Bureau of Mines engineers have been active in examining all reported finds, as there is real difficulty in bringing in imported gypsum at present on account of shipping difficulties, and the price of ore has more than doubled.

▶ The outstanding development in the Baguio gold district of Mountain Province during the year has been the expansion of the Big Wedge mine, controlled by Atok Gold Mining Co. Persistent and intelligently planned exploration work finally succeeded in bringing in an important orebody on the extension of the Keystone vein, which had proved disappointing when encountered on the upper levels. Exploratory work on the 800 level disclosed high gold content and further developments to the 1,200 level were successful in blocking out an orebody which, according to a statement by the management, has shown a length of over 900 ft., averaging 55 ft. wide, with an average gold content of 54.80 pesos per ton. Plans had been made late last year for doubling the size of the then 200-ton mill, and as orders had been placed for additional equipment before the shipping and priority situation became acute, construction was completed early in August. Since then the mill has functioned with complete success, with bullion output of approximately 500,000 pesos per month.



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Mobilize for Campaign Against Carelessness in Mining

Safety Council Congress deliberates on rising tide of industrial accidents—Inexperienced men working in mines must observe underground safety rules

ACCIDENTS which kill or disable workmen are increasing at an alarming rate in practically every field of industrial activity, including mining, the deliberations of the 30th National Safety Congress in Chicago, Oct. 6 to 10, disclosed.

Responsibility for the sharp rise rests partly on the shoulders of new, inexperienced men, rushed into jobs without adequate training in accident-prevention practices. But management, too, is blamed, because in the imperative need to produce at top speed for the national rearmament program, executives feel that time, money, and men can be better used than in promoting safety campaigns.

Col. John Stillwell, New York electric utility executive and president of the National Safety Council, in his opening address at the convention, revealed that from Jan. 1 to Sept. 1 of this year industrial accidental deaths are up 6 percent over 1940 losses and all accidental deaths are higher by 3 percent.

"We must contemplate, then," said Colonel Stillwell, "a possible accidental death toll for the entire year of 100,000 lives, or 3,500 more than were lost in 1940. The probable economic losses will be some $3,600,000,000. This situation is responsible for slowing up defense.

He proposed a three-point program to curb accidents; this to include organization of public support in an "Enlist For Safety" campaign; a study of new emergency problems to discover and apply appropriate remedies; and "mobilization of all our strength to defeat the pressure for diversion of safety budgets and manpower to other uses."

A. W. Macdonald, superintendent of industrial relations for the Dominion Steel & Coal Corp., Ltd., of Sidney, Nova Scotia, found many reasons for mine accidents, such as unsafe practices, poor discipline, mechanical hazards, and incompetency among employees. The Canadian practice, he said, is to keep a record of accidents under each boss.

A fear of being deemed a "panty waist," plus an ancient dread of placing things over the eyes, was blamed by C. E. McKnight, safety director of Lake Shore Gold Mines, Ltd., of Kirkland Lake, Ontario, for hindering efforts to persuade miners to wear goggles as protection against eye hazards.

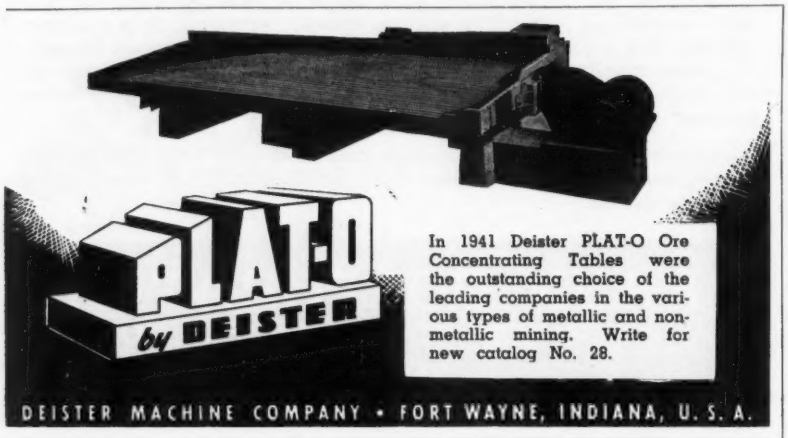
"Dated dynamite" was recommended as a safety measure by E. D. McNeill, Jr., of the Hercules Powder Co., Virginia, Minn. "Maintenance of fresh stocks of explosives," said Mr. McNeill, "is highly important to the safety as well as efficiency of mining operations. The powder should be rotated to insure fresh stocks always and each case of powder or dynamite should be marked with date of delivery to the mine."

He also urged the use of delayed electric-firing systems in all mine blasting operations, saying that in tunnel driving in recent years mile after mile of large tunnels has been driven with unheard-of speed and excellent safety records where such delayed electric-firing methods were used. Manufacturers of the firing devices, he stated, are supplying dependable products and operators are following safe firing programs. He also recommended a sound respect for lightning and cautioned against handling explosives during electric storms, even deep in the ground.

W. T. Winning, mine inspector for the Bituminous Casualty Corp., Herrin, Ill., declared that for the miner the "gun that wasn't loaded" is "the perfectly good mine roof that didn't need timber supports."

Mine inspectors, from their experience, he stated, are able to refute the popular belief that mining machinery is responsible for a large proportion of mine accidents. "It is not the machinery itself that causes accidents," he declared, "but carelessness with which it is handled and the hazardous conditions under which it is operated."

The exposition of industrial safety appliances staged in connection with the Safety Congress displayed everything from gas masks, goggles, gloves, to safety shoes, soap, salt, and sunlamps.



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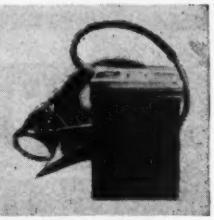

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EQUIPMENT NEWS

Priorities Order P-56 for Mining Is Consolidated and Simplified

Four types of priority ratings are available for worthy mining operations—A-3 rating given new machinery and to obtain repair parts as order P-23 expires Dec. 31.

THE FOLLOWING INFORMATION was obtained early in December from direct discussions with OPM mining branch officials in order to supply readers with the latest changes made in mining priorities Order P–56.

Mining priorities are now consolidated under a single priorities order of OPM, and are administered by a special OPM division headed by Dr. Wilbur A. Nelson. Not only repair and maintenance equipment and operating supplies, but also new equipment for expansion has been consolidated in a revised P–56 priorities order, issued on Dec. 1.

The new set-up retains the state coordinator staff and the individual mine serial number plan of the original P–56 order. Mines which already have serial numbers retain them but operate under the amended order; those without serial numbers which are eligible for priority assistance should apply to the State coordinators for certification as in the past. (A list of these coordinators appears on the following page):

Under the new P–56 order, four types of priority aid are available to worthy mining operations. By worthy ventures, OPM means those which are contributing metals, minerals, and coal needed in national defense. These are the types of assistance available:

Rating A–3 or A–8, for obtaining a steady flow of needed materials and equipment for repair and maintenance and certain operating supplies. The A–3 rating is available for repair for mining machinery listed under Schedule A of the amended order, which includes about 65 classes of machinery. The A-8 must be used for all other repair and maintenance and for operating supplies. These ratings are assigned by mining officials themselves to orders on suppliers. Orders must be accompanied with the mine's serial number, and the use of the rating must be reported monthly on Form PD–119 to the state coordinator.

Rating A–1–c, for obtaining materials and equipment necessary for emergency inventory. Application for this rating must be made directly to Dr. Nelson's staff in Washington and is directed to the ultimate supplier. Separate ratings are required for each specific purchase. Each authorization for use of this rating is given a number known as "authorization number" and must not be confused with the serial number permanently assigned to each mine.

Rating A–1–a, for obtaining materials for repairs in cases of emergency breakdown. Procedure for obtaining this rating is the same as for A–1–c.

Rating A–3, for obtaining *NEW* equipments for expansion of operations or for starting a new venture, unless its magnitude warrants a project rating. Again, the rating must come from Washington, but unlike the A–1–a and A–1–c ratings, the A–3 rating is extendable down the production line from the mine making the purchase through the manufacturer to his suppliers and sub-suppliers.

Inclusion of the A–3 rating for new machinery and permission to use an A–3 for obtaining many repair parts are the biggest changes made by the issuance of the revised order. The new machinery priority previously has been available only to certain manufacturers under P–23 (now expiring Dec. 31); switching the priority to the user means that mine operators may designate anyone they desire to produce their equipment, thus giving small neighborhood shops a chance to stay in business. The change also gives Washington specific control over the ultimate purpose for which mining machinery is to be used.

As to all machines listed below, the rating A-3 provided herein likewise applies to equipment items, accessories, and tools customarily sold with such machines.

Schedule A

Aerial tramway equipment
Air compressors for mine use
Air distribution equipment
Assaying and testing laboratory equipment at the mine
Ball-casting machines
Box-car loaders
Cages and skips
Car-dumpers—rotary or end
Cleaning plants and equipment, including design, construction, and erection
Concentrating plants and equipment, including design, construction, and erection
Conveyors—shaking, belt, or chain types, including duckbills and other self-loading heads
Cutting machines—cable reel and self-propelling transportation trucks therefor
Diamond core drilling machines
Dragline dredges, excavators, and scraper units
Dredges—continuous bucket
Drills and drilling machines, power-driven, and reconditioning equipment therefor
Dust-control equipment
Electrical equipment for mine transportation and power
Hoists—including room hoists and car pullers
Hydraulic monitors
Jacks for lifting and roof support
Lamps—Mine, miners, safety, and ore-exploration types
Locomotives for mine use
Loaders, mobile, including mucking machines
Milling plants and equipment, including design, construction, and erection
Mine cars, track or trackless
Pit-car loaders and elevating conveyors
Preparation plants and equipment, including design, construction, and erection
Pumps, pipe, and fittings for mine drainage or material transport
Rock-dusting equipment
Sand dryers
Scraper loaders
Sheaves and sheave blocks
Shovels, power
Shuttle cars, track or trackless
Slusher hoists and scrapers
Storage batteries for mine use
Tanks and bins for storage of mine products
Tipples and headframes, including design, construction, and erection
Track and track accessories for mine transportation
Treating plants and equipment, including design, construction, and erection
Trucks, tractors and trailers for mine use
Ventilation equipment
Waste disposal equipment
Weighing equipment, including automatic devices

The new arrangement establishes the OPM mining division as an independent section working in close harmony with each of the materials branches of the defense agency which are charged with supply and demand problems of metals and minerals. Under this plan, it is expected that use of the PD–1 form for priority aid will be dispensed with in all except relatively isolated cases of extreme hardship.

Mine Operators Must Be Specific

Coincident with issuance of the revised order, Dr. Nelson urged mine operators to be specific in applying to Washington for assistance. Most of the delays in meeting priorities problems are due to a lack of sufficient information. Here is a simple set of rules to follow:

If it's about getting a serial number, address your State coordinator and give him the fullest information about your operation. The application should list the name of the company, location and post office address of each mine, the metals produced, average monthly production of metals, and number of employees.

If seeking information or starting a new venture, address Dr. Nelson in Washington (Dr. Wilbur A. Nelson, Administrator, Mine Priorities, Office of Production Management, Washington, D. C.). Be sure to state your serial number and give full data.

If applying for an A–3 rating for new equipments, in addition to the rules in the preceding paragraph, be sure to state why you need the equipment in terms of the production which will result.

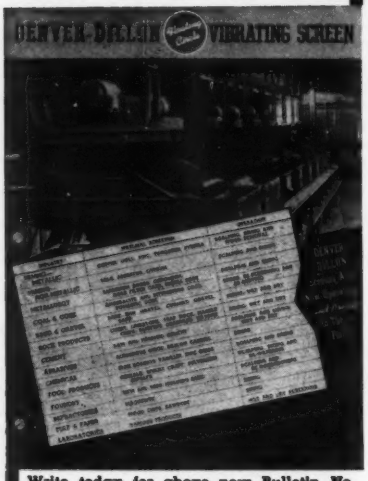
If applying for an A–1–c rating, indicate your past inventory procedure and why the materials or equipment is required for emergency inventory now. An illustration would be a sand pump impellor, which you know must be replaced, say, every 30 days.

If applying for an A–1–a rating, give full information as to what has happened, and why you need help.

In instances involving either A–1–a or A–1–c, give the ultimate supplier of the article needed; if your local supplier must obtain it, find out and state from whom and give that supplier's name. Neither of these ratings can be extended beyond the supplier upon whom they are issued.

A final rule is that the more detailed

A NEW BULLETIN YOU WILL WANT



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State Coordinators have furnished OPM with a list of active mines in their states, and are available for consultation or to carry out other services for Washington. Cases involving mining companies in the doubtful category for priority rating are being discussed carefully, and supplementary lists may be submitted at any time by State Coordinators to Washington. Reclaiming of tailings, dumps, sulphur, and salt mining may be considered samples of doubtful cases.

Gold Placer Mining (all types of dredging, hydraulicking) is not included in Order P–56. This type of operation may be included, however, if other metals are recovered. Gold placer operators may, however, obtain priority assistance by the use of preference rating order P–22, which is a general priority plan for industry repair and maintenance.

Gold lode mining is within the scope of P–56.

Exploration of ore bodies adjacent to operating properties is considered within the order.

Prospecting for essential and necessary materials will be encouraged and the Administrator may assign serial numbers to individual prospecting ventures.

Quarries are covered; so are sand and gravel operations and, in the new order, refractories.

Leasing—Properties operated under contract or by lessees may use the serial number of the mine in buying supplies and repair parts, if so desired and if the mine operator will assume full responsibility for compliance with the order.

Smelting—The scope of the order ends at the beneficiation, concentration, or preparation for shipment of the products of metal and non-metallic mining, but does not include smelting and refining of metals, or final extraction of minerals or metals.

Mines which cannot obtain a serial number under provisions of P–56 may obtain repair and maintenance equipment under P–22, issued Sept. 9, which permits them A–10 rating.

Foreign mining companies, at the discretion of the Director of Priorities, may be issued a serial number or numbers to a person operating a mining enterprise outside the limits of the United States, its territories and possessions.

State Emergency Coordinators for Mines

(A preliminary survey made by *E.&M.J.* indicates all State Coordinators are not receiving form PD-119 that all mining companies having a serial number must fill out and mail by the 10th of each month.)

State	Coordinator
Alabama	E. J. McCrossin, Phoenix Building, Birmingham
Arizona	C. M. Martin, 1821 East Jackson St., Phoenix
Arkansas	George C. Branner, 447 State Capitol, Little Rock
California	Kenneth R. Fulton, Ferry Building, San Francisco
Colorado	Robert S. Palmer, Colorado Mining Association, 204 State Office Building, Denver
Connecticut	George L. Burke, Commissioner, Dept. Public Works, Hartford
Delaware	Desmond Lyons, Tax Commissioner, 843 King St., Wilmington
Florida	Herman F. Gunter, Box 631, Tallahassee
Georgia	Capt. Garland Peyton, State Capitol Building, Atlanta
Idaho	C. E. Arney, State Capitol Building, Boise
Illinois	Robert M. Medill, State Capitol Building, Springfield
Indiana	Clarence A. Jackson, 612 Board of Trade Building, Indianapolis
Iowa	George Duckworth, State Capitol Building, Des Moines
Kansas	Jeff Robertson, 801 Harrison Street, Topeka
Kentucky	Moss G. Patterson, Dept. of Mines & Minerals, Box 680, Lexington
Louisiana	Joseph L. Micaugh, Division of Mines, State Capitol Building, Baton Rouge
Maine	Ralph E. Ranger, Industrial Engineer, State House, Augusta
Maryland	John J. Rutledge, State Capitol Office Building, Annapolis
Massachusetts	Dr. Warren J. Mead, Professor of Geology, Mass. Institute of Technology, Cambridge
Michigan	Benjamin F. Sparks, State Highway Department, Escanaba
Minnesota	Ray Nolan, Hibbing
Mississippi	Col. Lea B. Robinson, Defense Council, Jackson
Missouri	John A. Skinner, Mine Inspector, Bureau of Mines, Jefferson City
Montana	Tom D. Caverly, State Capitol Building, Helena
Nebraska	Blain Yoder, State Capitol Building, Lincoln
Nevada	Matt Murphy, State Capitol Building, Carson City
New Hampshire	Edward Ellingwood, State House Annex, Concord
New Jersey	Meredith E. Johnson, Room 417, State House Annex, Trenton
New Mexico	John M. Kelly, State Capitol Building, Santa Fe
New York	Miss Freida S. Miller, 80 Centre St., New York City
North Carolina	Dr. J. L. Stuckey, State Capitol Building, Raleigh
North Dakota	W. T. Kraft, 315 Broadway, Bismark
Ohio	George Strain, Department of Industrial Relations, State Capitol Building, Columbus
Oklahoma	J. G. Puterbaugh, McAlister
Oregon	Earl K. Nixon, Director, State Department of Geology and Mineral Industries, State Capitol Building, Salem
Pennsylvania	Hon. Richard Maize, Commonwealth of Pennsylvania, Dept. of Mines, Harrisburg
Rhode Island	Frank A. Sullivan, P. O. Box 502, Westerly
South Carolina	J. Roy Jones, Commissioner of Agriculture, Wade Hampton Building, Columbia
South Dakota	Hon. Harry H. Stewart, Deadwood
Tennessee	S. E. Bryant, State Capitol Building, Nashville
Texas	Dean John W. Kidd, College of Mines & Metallurgy, El Paso
Utah	Wendell Grover, Industrial Commissioner, State Capitol Building, Salt Lake City
Vermont	William L. McKee, Unemployment Compensation Commissioner, Montpelier
Virginia	Thomas B. Morton, State Capitol Building, Richmond
Washington	Robert H. Harlin, State Director, Labor & Industry, Olympia
West Virginia	N. P. Rhinehart, Chief, State Dept. of Mines, Charleston
Wisconsin	A. H. Fineisen, State House, Madison
Wyoming	J. M. Sampson, Rock Springs
Alaska	Benjamin D. Stewart, Territorial Commissioner of Mines, Juneau
Philippine Islands	Robert J. Hoffcut, economic assistant to High Commissioner, Manila
Puerto Rico	Martin Lopex Sanabria, Dept. of Interior, San Juan

Each mine having obtained a serial number has been furnished with a sample form PD–119, upon which is reported the quantity of equipment or supplies purchased each month and the name of the supplier. These reports are to be filed each month with the Coordinator in each State, and will be subject to review by an authorized agent of OPM. Copies of all monthly reports must be segregated and retained by each mine and all orders for materials and supplies placed under the rating must be filed apart from other orders. Purchases under the common form of finished steel, electrical equipment, and copper wire can be grouped together.

OPM has instructed mine operators to conform to the inventory standard established; not to exceed a ratio of inventory to output in quantity terms that prevailed at the year end of 1938, 1939, and 1940, or the respective fiscal year end, but always to be kept at a reasonable minimum. Small items and their ratios need not be computed. From such an inventory set-up operators will have a yardstick or standard of the important classes of supplies and equipment required to maintain production.

In addition to Dr. Nelson, the OPM staff administering the P–56 order includes Dennis K. Pickens, assistant administrator; Dennis L. McElroy, technical advisor on coal, and John L. G. Weysser, assistant to McElroy. Technical advisors and assistants on other metals and minerals will be added to the staff as they are needed and available.

Battery-Operated Mine Locomotive

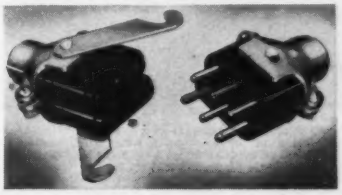
FOR USE in coal, metal, and non-metallic mines, a new storage-battery tramming locomotive is announced by Westinghouse Electric & Mfg. Co., East Pittsburgh, Pa. Known as the Midget and weighing only 1.5 tons, the locomotive is operated by a 5-h.p., 40-v. d–c traction motor. Continuous motor rating is 770

r.p.m., 55 amp., 40 v., with a gear ratio of 11⅔ : 1. Chassis is of heavy, one-piece-, cast-steel construction. Frame is spring suspended for cushioning of road shocks. Power is transmitted through torsion ring coupling to worms driving worm gears. The rated drawbar pull is 400 lb. The coupling carries the weight of the armature and aligns it with the driving worms. Locomotive brake is of the band type operating directly on a drum mounted on an extension of the worm gear shaft. The drum fit on the shaft is of the tapered type for easy removal. Brake operation is by a handle set at an angle to permit maximum braking with the least effort. A ratchet holds the brake in fixed position when necessary. A three-position controller is arranged for series-parallel control of the motor fields in either direction.

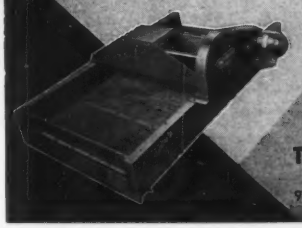
6-Conductor Trailing Cable Plug

OHIO BRASS CO., Mansfield, Ohio, announces development of a new six-conductor Mechano-Plug for effecting trailing cable connections. Designed for making connections to reversible motors, terminals of new plug accommodate No. 6 and smaller sizes of wire.

The new Mechano-Plug, as its name implies, is a mechanically assembled and installed plug. Eliminating vulcanizing or unsightly and dangerous cable splices, it can be installed in the field with the simplest tools. Simply strip the insulation from the end of the cable leads, insert, and clamp the individual wires into the bronze terminal plugs or sockets, and screw same into position in the



THE Leahy HEAVY DUTY
No-Blind VIBRATING SCREEN with TRI-VIBE —for Fine Mesh Screening



Prevents Blinding
Screens Faster and Better
Saves on Power

Tri-Vibe Equipped Type C Leahy Screens are guaranteed to operate with only ½ H.P. and an annual maintenance less than 1% of purchase price.

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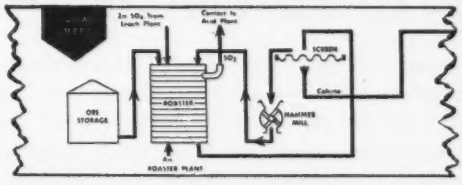

ROASTING
CALCINNING
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"ST. JOE" *zinc oxides are produced with Nichols Herreshoff Multiple Hearth Furnaces, the first step in a long series of operations.*

Definite superiorities in design and construction have earned Nichols Herreshoff Multiple-Hearth Furnaces a long standing, and unexcelled, reputation for all around excellence —regardless of the type of service. As have other users, you, too will quickly recognize that with a Nichols Herreshoff Multiple Hearth Furnace you can do the job better—and at less cost. Look into the specific advantages our equipment gives you. Get the benefit of our specialized experience. Ask for recommendations on your particular problems or write for Bulletin 206.

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UNIVERSITY TOWER
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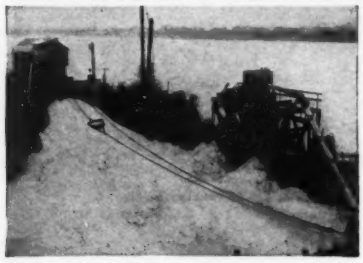
Bulk Storage Simplified

Above is a typical example of a Sauerman Scraper installation of large capacity with self-propelled tail tower as used at mines and processing plants for open storage of materials. This type of machine is able to cover a wide storage area and handle a large tonnage both storing and reclaiming.



A small Sauerman Scraper stockpiles crushed silica rock on an irregular ground space alongside a washery and reclaims this stored material as required to a drying plant.

with an economical SAUERMAN SCRAPER

At mines, mills and smelters, Sauerman Power Scrapers are being used most effectively in the stockpiling of both raw and processed materials, either in the open or inside buildings.

One of the great advantages of this equipment is its flexibility, which permits Sauerman engineers to adapt each scraper installation to the exact needs of the job.

Moreover, the first cost of a Sauerman installation is reasonable, maintenance amounts to very little, and the simplicity of operation makes it possible to place the control of even the largest installation in the hands of one operator.

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More Mining Men Are Buying Today!

1,500 more key men are paying to read E. & M. J. today than were on our rolls at this time last year . . . And 8.4% of these readers have been added since January of this year!

It seems trite to say that the metal and non-metallic mining fields and the men who operate its mines and plants need more information today than ever before. But it is true—and because of this E.&M.J. is playing a more vital role in keeping these men posted on up-to-date methods and equipment for expanded metal and mineral production.

rubber insulating body of the plug. Then tighten the malleable iron clamping frame. The Mechano-Plug is said to be installed in the field without utilizing special tools or equipment. A color key simplifies proper polarity hook-up at the time of installation. The plug body is of rubber, which provides ample insulation against electric shock. Moisture and dust-proof joints insure a positive, leak-proof electrical connection. The six-conductor Mechano-Plug is designed for currents of 80 amp., voltages from 250 to 600 d.c.

The company also recently announced the development of a new plier-type ground clamp for effecting ground- or return-circuit connections to rail-bond cable or rail. Known as the Junior Plier-Type Ground Clamp, it has ample capacity to handle ground connections for drills and similar equipment.

New Babbitt Metal

A NEW BABBITT METAL for bearings subject to high pressures and temperatures has been announced by the Magnolia Metal Co., 120 Bayway, Elizabeth, N. J. The metal is called power nickel genuine babbitt and is said to have a tensile strength of 17,500 lb. per square inch, a yield point of 65,000 lb. per square inch, a Brinell hardness of 27, and its pouring temperature ranges from 950 to 1,000 deg. F. Its strength is said to make it adapted to heavy bearing loads.

BULLETINS

Belt Conveyors. Goodman Manufacturing Co., Halsted St. at 48th Place, Chicago, Ill. Bulletin C-411 describes and illustrates the manufacturer's belt conveyors for use underground. Pp. 23.

Screw Classifiers. Western Machinery Co., 760 Folsom St., San Francisco, Calif. Bulletin SC-941 describes and illustrates features of the manufacturer's screw classifiers. Specifications and drawings of the various sizes are also included in addition to tables of engineering data. Pp. 14.

Vibrating Screen. Denver Equipment Co., Denver, Colo. Bulletin S3-B3 describes and illustrates the company's new Denver-Dillon vibrating screen. Pp. 4.

Pumps. Kimball-Krogh Pump Co., 301 West Avenue 26, Los Angeles, Calif. Bulletin 150-A describes and illustrates features of the manufacturer's Type NKE 2-stage high-pressure multiple-duty pumps. Pp. 4.

Rail Clamps. Robins Conveying Belt Co., Passaic, N. J. Bulletin 114 describes and illustrates features of the company's rail clamps for movable structures used in loading and unloading of materials. Pp. 17.

Crushers. Allis Chalmers Mfg. Co., Milwaukee, Wis. Bulletin B-6006-B describes and illustrates the company's Type R reduction crusher having a speed-set product-control operation. Numerous sectional views and installation photos are included. Pp. 16.

Molybdenum in Steel. Climax Molybdenum Co., 500 Fifth Ave., New York, N. Y. The company recently issued its latest and revised sections containing data on "Molybdenum in Steel." The new sections added include data on S-curves, hardenability, and microstructure studies of molybdenum steel, carbon molybdenum steels, molybdenum tool steels, molybdenum high-speed steels, physical property charts, and a revised general index.

Magnets. Dings Magnetic Separator Co., 526 E. Smith St., Milwaukee, Wis. Catalog 301 describes features of the manufacturer's latest type of electric double-gap magnets. Specifications for the various types of magnets and installation photographs are given. Pp. 9.

Pumps. Pomona Pump Co., 206 E. Commercial St., Pomona, Calif. Bulletin 29 contains practical information concerning pump adaptation for a wide range of duties under varying operating conditions. The brochure is well illustrated and shows varied applications of the manufacturer's industrial pumps. Pp. 23.

Drifter Drill. Gardner-Denver Co., Quincy, Ill. Bulletin CF-1 describes and illustrates features of the company's new continuous-feed CF-89-H drifter drill. Pp. 4. Bulletin D-73 describes and illustrates the company's new D-73 2¾-in. hand-rotated drifter drill. Pp. 4.

Portable Air Compressors. Schramm, Inc., West Chester, Pa. Catalog 42-P describes and illustrates the company's line of portable gasoline- and diesel-engine driven compressor units. Pp. 19.

INDUSTRIAL NOTES

Taylor Wharton Iron & Steel Co. recently placed in operation a new Pittsburgh Lectromelt furnace at its Highbridge, N. J. plant. This installation will be an important factor in conserving ferromanganese steel in the manufacture of Hadfields Manganese Steel. This new unit places operations at the Highbridge plant on a 100 percent electric steel-making basis.

The Du Pont Company recently opened its new Cyanide Products Service Laboratory at the Niagara Falls, N. Y., plant of The R. & H. Chemicals Department. Staffed by chemists, metallurgists, and technical service men, the company reports the new laboratories will be used primarily for the study and solution of customers' problems and for development of improved metal-treatment and electroplating processes.

Grinding Pebbles and Tube-Mill Liners

SINCE the spread of the European war virtually cut off imports from Denmark, Belgium, and France, the Bureau of Mines has had numerous inquiries as to domestic sources of grinding pebbles and silex or other type of ball-mill liners.

Uses—Flint pebbles and tube-mill liners are used by grinders of certain ceramic materials to prevent contamination of the finished product by iron abraded from steel balls and liners. The feldspar industry is one of the largest users of flint grinding mediums; they are employed also by certain grinders of talc and soapstone, ground sand and sandstone, quartz, and chalk. Some paint materials, such as lithopone, have been ground in pebble mills. Flint pebbles have been used more or less in the past, especially by cement manufacturers, and for fine-grinding of gold, copper, or other ores in which iron contamination is not especially deleterious.

Imports—Grinding pebbles have been imported from France and Denmark and silex liners from Belgium. It is impossible to state the exact tonnages of grinding pebbles and liners imported, as other pebbles and flints of various sizes and shapes intended to be ground for use in ceramic bodies are included in the customs classification of "flint, flints, and flintstones." For some years before 1930, pyrophoric alloy "flints" for pocket lighters and gas lighters likewise were included. During the past 15 or 20 years, however, imports under this more general classification have ranged from 26,636 short tons in 1920 to 3,754 tons in 1932, and for 1922 to 1929, inclusive, they ranged from 14,000 to 17,000 tons. After the 1932 low, imports increased gradually to 13,428 tons in 1937 and were 11,987 tons in 1939. Since the first quarter of 1940, imports undoubtedly have been greatly curtailed and perhaps have been nonexistent.

Only two firms, one Danish and one French, have supplied foreign pebbles to American consumers. The pebbles occur in distinct beds in the white chalk cliffs of the islands of Zealand, Möen, and Langeland, on the Danish coast, and on the French coast, largely between Havre and Dieppe. They are gathered by hand along the beach, washed with water in large rotating drums, then run over a picking belt where "fine-quality blue nodule" flint and "prime-quality hard" flint pebbles are picked out separately, leaving second-quality flint and granite for construction work or waste. In France, crude pieces of flint of any size or shape and the less-tough pebbles also are gathered and exported for use as crushed flint in potteries. Although no universal standard for grading flint pebbles is recognized, the accompanying classification of Danish pebbles into seven grades according to size has been used by one importer:

Classification of Danish Pebbles

Size No.	Diameter, in.
1	1 to 1¾
2	1¾ to 2⅝
3	2⅝ to 3¼
4	3¼ to 4½
5	4½ to 5½
6	5½ to 6¾
7	6¾ to 7⅞

An analysis of French flint pebbles gathered near Dieppe follows:

	Per Cent
Silica	98.20
Alumina	0.30
Chalk	0.40
Magnesia	0.10
Loss on ignition	1.00
	100.00
Specific gravity	2.76

Germany and the United States have been the two leading markets for Danish pebbles, accounting for 60 percent of the total exports in recent years. Other markets included Norway, Sweden, Australia, Brazil, and Japan.

Belgian silex lining was quarried like building stone, and the larger pieces were rough-cut or shaped by hand chisels and hammers somewhat like large granite paving blocks. According to a recent informant, the sizes formerly supplied from Belgium were 2, 2½, 3 and 4 in. thick; usually between 4 and 5 in. wide; and from 6 to 9 in. long.

Domestic Production—For several years the only commercial production of grinding pebbles and tube-mill liners known to the Bureau of Mines has been that of the Jasper Stone Company, Sioux City, Iowa. This firm has marketed mill liners and artificially rounded pebbles from its quartzite deposits near Jasper, Rock County, Minn., for a long time. In the past it produced cut cubes, mill liners, and pebbles from the Sioux quartzite in Iowa and near East Sioux Falls, Minnehaha County, S. D., but no production from these localities has been reported for 10 to 15 years.

Before 1936 a steady output of beach pebbles from the shore line between Oceanside and Encinitas, San Diego County, Calif., was reported. These pebbles, largely fine-grained tuffs, porphyry, and granite, were washed up on the beaches after severe storms, entangled in seaweed torn loose from the ocean floor. They ranged from 2 to 7 in. in diameter; were smooth, round, or nearly round; and were picked by hand from the beaches, sorted roughly according to size, and used chiefly in grinding cement. Although tough and said to be harder to break than Danish flint pebbles, the California pebbles wore somewhat faster on hard material, being coarser-grained and less smooth than the Danish pebbles. On very hard ore they were considered to be about half as efficient and required a larger charge; however, they cost less than half as much.

Various types of selected, well-rounded stream or beach pebbles of rhyolite, basalt, granite, or other hard, tough rock have been used locally, particularly in the Rocky Mountain region and in the Pacific Coast States. Dense, banded rhyolite blocks rounded by tumbling formerly were obtained near Manhattan, Nev., and used in metallurgical mills. Granite pebbles found in the Sierra Nevada Mountains are used by some gold mills. Chunks of flinty ore and tough blocks of trap rock have been utilized as grinding mediums for tube mills. The United States Reclamation Service used angular blocks of basalt rock in its cement mill during the construction of the Arrowrock dam; the corners and edges were quickly ground off, and after the blocks were reduced to spherical or ellipsoidal forms, the material was said to wear better and longer than Danish pebbles.

Possible Domestic Sources—Pebbles of granite, quartzite, and other stone have been imported from the Canadian shore of Lake Superior and from Conception Bay, Newfoundland, for use in cement mills. Rhyolite and felsite pebbles are abundant along the New England seashore from Marblehead, Mass., to Eastport, Me. Some of these pebbles are quite tough, but samples have been rejected for feldspar grinding because of the iron content. In the Gulf States and in the Southern Appalachian and Ozark Mountain regions at the edge of the Coastal Plain are numerous deposits of flint pebbles, most of which do not exceed 2½ in. in diameter, along stream terraces and bottoms. The Cretaceous limestones in central Texas, particularly the Edwards limestones, are flint-bearing. Other tough pebbles of siliceous materials, such as agate, carnelian, or petrified wood, found in abundance in certain parts of the country, might be suitable as grinding mediums. Extensive deposits contain-

ing pebbles of novaculite, a very hard and dense siliceous rock in Pike, Howard, and Sevier Counties, Ark., have been described by the Geological Survey.[1]

Tube-mill liners and possibly artificially rounded grinding pebbles formerly were supplied from the Ocala limestone beds in Florida and from the Ft. Payne chert near Iron City, Tenn.[2] Mill liners and artificially rounded pebbles also might be obtained from one of the quartzite belts of central and southeastern Pennsylvania or from the ganister or quartzite rock of Calhoun, Talladega, Cherokee, Clay, and Cleburne Counties, in Alabama.

Recently the Bureau of Mines has been informed that the United Feldspar & Minerals Corporation, Minpro Division, Spruce Pine, N. C., has experimented with quartzite from its Deer Park property and with topaz for both mill liners and grinding pebbles with highly satisfactory results. Quartz fragments used for making potters' flint are withdrawn from the mills as soon as they have become sufficiently rounded and then used as pebbles in the feldspar grinding mills.

Information as to undeveloped deposits in various states usually can be obtained from the respective State geologists. The following firms have indicated that they may be able to supply quartz pebbles or liner blocks:

Dealers, importers, and suppliers of grinding pebbles and silex liners:

California: Atkins, Kroll & Co., 200 California St., San Francisco; King Solomon Gold Mines, Black Bear (or Crocker Bldg., San Francisco); John C. Momand, Carlsbad.

Delaware: E. I. Du Pont de Nemours & Co., Inc., R. & H. Chemicals Dept., Dupont Bldg., Wilmington, Del.

Iowa: Jasper Stone Co., Sioux City, Iowa.

Maryland: Clinchfield Sand & Feldspar Corporation, Hearst Tower Bldg., Baltimore.

New Jersey: Paul O. Abbe, Inc., 130 Center Ave., Little Falls; Eureka Flint & Spar Co., Trenton.

New York: Abbe Engineering Co., 54 Church St., New York; J. R. Alsing Engineering Co., 50 Church St., New York; Buedendorf Bros., 2 Broadway, New York; John W. Higman Co., 24 State St., New York; New Jersey Pulverizing Co., 205 West 34th St., New York; F. L. Smidth & Co., 225 Broadway, New York; Taintor Trading Co., Inc., 21 State St., New York; B. Voigt, 250 W. 57th St., New York.

North Carolina: United Feldspar & Minerals Corporation, Minpro Division, Spruce Pine.

Ohio: Patterson Foundry & Machine Co., East Liverpool.

Pennsylvania: Philadelphia Steel & Iron Co., Washington Ave., Conshohocken; Traylor Engineering & Mfg. Co., 666 South 10th St., Allentown; Charles A. Wagner Co., Inc., 811 Callowhill St., Philadelphia.

Tennessee: J. Howard Swaim, 916 Cotton States Bldg., Nashville.

Wisconsin: Manitowoc Engineering Works, Manitowoc.

Reprinted from Information Circular 7139, by Robert W. Metcalf, U. S. Bureau of Mines, Washington, D. C.

[1] Miser, H. D., and Purdue, A. H.: "Gravel Deposits of Caddo Gap and DeQueen Quadrangles, Ark." Geol. Survey Bulletin 690, 1919, pp. 15–30; "Geology of the DeQueen and Caddo-Gap Quadrangles, Ark." Geological Survey Bulletin 808, 1929, 195 pp.

[2] Miser, Hugh D.: "Mineral Resources of the Waynesboro Quadrangle, Tennessee." Tennessee State Geological Survey, Bulletin 26, Nashville, Tenn., 1921, pp. 132–135.

Metal-Mining Research In 1940

THE principal problems under investigation in 1940 by the Metal-Mining Research Section (McHenry Mosier, Supervising Engineer), of the U. S. Bureau of Mines, were (1) the development of a method for the determination of pressures on rock pillars in underground mines, (2) stemming in metal mines, (3) dust studies in rock drilling, and (4) alloy steels for rock drilling. All of the investigations except the first were conducted at Mount Weather testing adit, Mount Weather, Va. The first is summarized as follows:

Determination of Pressure on Rock Pillars in Underground Mines—Although a method of determining loads on pillars in underground mines long has been wanted and needed by the mining industry, the current scientific approach by the Bureau of Mines is believed to be the first to give promise of ultimate success by methods readily applicable under usual underground conditions.

Primarily, measurement of pressures in mine pillars will result in greater safety for miners by directing attention to dangerous conditions the existence of which it would be impossible to detect otherwise. Conversely, the measurement of pressure on ore pillars after partial extraction of an ore deposit will disclose which pillars are carrying the load and which pillars may be removed without disturbing the natural arches in the surrounding rock.

Three sonic methods for the solution of this problem are now being developed: (1) velocity of induced vibration, (2) damping of induced vibration, and (3) sub-audible noises. The velocity method is based on the increase in velocity of sound through rock with increase in pressure. First, the pressure-velocity relationship is determined at known pressures in the laboratory from representative samples taken from a rock pillar under test.[1] Then, the velocity of sound is measured in the pillar in situ.[2] A method and apparatus for measuring this velocity with an error of less than 1 percent have been developed. The measurement is obtained with the aid of a high-speed oscillograph, which records the time required for a sound wave to travel a known distance through the pillar. Either a percussion blow or an electric blasting cap may be used for the wave excitation. It has been demonstrated that this method applies to rock pillars whose sound absorption in the medium is small and whose length is great compared with the radius. Discontinuities in the rock do not affect the velocity.

The applicability of this method for measuring the elasticity of concrete pillars in situ has been demonstrated. In connection with field work at lead mines in Southeast Missouri, the elastic constants measured in situ in concrete pillars constructed to replace ore pillars and those measured in laboratory samples of the concrete agreed within the limits of the experimental error.

The damping coefficients of rock and of concrete are being studied to ascertain if it is possible to determine pressure in rocks by measuring this physical quantity. Laboratory methods have been developed for obtaining this coefficient, and it has been determined for numerous samples of rock and of concrete.

It is planned to measure pressure on mine pillars and arches for several systems of mining in various formations to ascertain the practicability and limitations of the methods. Field work at the close of the year was progressing favorably in the copper mines of northern Michigan. Particular attention will be paid to pillars in deep mines, where the pressure may become great enough to cause a rock-burst or failure of a mine pillar with explosive violence.

In both the Tri-State zinc district and the Copper Peninsula of northern Michigan, sub-audible noises have been detected in rock pillars through high-gain amplifiers. Further work is necessary to interpret this phenomenon and determine its significance.

Excerpted from Report of Investigation 3536, Annual Report of the Mining Division, Fiscal Year, 1940, U. S. Bureau of Mines, Washington, D. C.

[1] Obert, Leonard: "Measurement of Pressures on Rock Pillars in Underground Mines, Part I." Bureau of Mines Dept. of Investigations 3444, 1939, 15 pp.

[2] Obert, Leonard: "Measurement of Pressures on Rock Pillars in Underground Mines, Part II." Bureau of Mines Rept. of Investigations 3521, 1940, 11 pp.

Kimberly Methods of Block-Caving Using Slusher Drifts

THIS PAPER is one of a series being published by the Bureau of Mines on mining methods and costs. It describes briefly the manner in which slusher hoists and scrapers are used in connection with block-caving methods at the Emma Nevada mine of the Consolidated Coppermines Corporation, Kimberly, Nev.

Kimberly is in the Robinson mining district, White Pine County, 7 miles west of Ely. The district is served by the Nevada Northern Railroad, which branches from the main line of the Southern Pacific at Wells, Nev., 140 miles north. Paved U. S. Highway 50 from Salt Lake City, Utah, to Reno, Nev., passes through Ely. The altitude at the collar of the Emma Nevada shaft is 7,117 ft. Surrounding hills are covered with a sparse growth of juniper, pinon, and mountain mahogany.

All mine timber and other supplies are brought in from a distance.

Acknowledgment is made to Paul J. Sirkegian, assistant superintendent and chief engineer; Mark A. Smith, general mine foreman; and others for helpful suggestions and cooperation.

Orebody—The principal orebody of the Consolidated Coppermines Corporation, which is serviced by the Emma Nevada shaft, is a gently undulating, south-dipping blanket of monzonite porphyry with a mineralized thickness of 60 to 400 ft.; the average is about 175 ft. Overlaying waste material is 200 to 500 ft. thick. The orebody proper is about 3,500 ft. long in an easterly and westerly direction and attains a maximum width of 1,200 ft. The copper mineral is chalcopyrite, which is distributed evenly through the mineralized area. The orebody is thoroughly fractured and altered and remarkably soft, which permits mining by a modified block-caving method. Virtually all the mineralized porphyry is underlain by a younger monzonite porphyry locally known as peanut porphyry, which in turn rests on a basement of sediments, either limestone or shale. The younger porphyry is fractured and altered less and generally stands better than the upper mineralized porphyry, although mine openings require heavy timbering in both cases. The waste material overlying the bodyore is unmineralized porphyry, sedimentary pendants, and very irregular bodies of post-mineral rhyolite, which often intrude and may underlie or overlie the ore. The northern limit of the orebody as explored to date is formed by a zone in which diminution of copper values in the porphyry renders further mining uneconomical. The ore terminates to the south against the intrusive rhyolites. Shaft sinking began in the fall of 1926, when existence of approximately 7,000,000 tons of ore was established.

Methods of Exploration—Most of the orebody was explored by churn-drill holes from the surface with samples taken at 5-ft. intervals. The churn-drill holes usually were 200 ft. apart, although the spacing of holes was governed partly by geologic study and partly by surface topography. Exploration by churn drills has been augmented to some degree by prospect headings from underground workings, short drifter drill holes, and diamond drilling. These latter methods have been used only to prospect fringe areas and to establish the location of irregular portions of the orebody.

After the churn-drill holes were completed and the top and bottom limits of the ore became known, contour maps of the orebody were prepared, showing the conformation of the top and bottom of the ore. The mining layout is based upon study of the contour maps and the sections built up from them.

The Emma Nevada shaft is in the footwall of the orebody at a sufficient distance therefrom to insure against caving of the shaft and plant structures as mining progresses. The haulage level was driven at an elevation that averaged about 60 ft. beneath the bottom of the orebody. Early mining followed the conventional block-caving method, using branch raises, which continues to be employed where particularly applicable. The orebody is mined in one lift, and the 60-ft. interval left between the haulage level and bottom of the ore was established as the most economical distance for development purposes. A greater distance to the ore from the haulage level would mean less drifting but more raising; on the other hand, if a smaller distance were chosen, less raising and a greater footage of haulage drifts would be required. Although the ideal distance from ore to haulage drift is 60 ft., the bottom of the orebody undulates and often is step-faulted, causing this interval to range from zero, where the body dipped down to the level, to a maximum height of about 130 ft. above the level. Such extreme variations, however, were rare and limited to relatively small areas.

The spacing of main haulage drifts and design of the branch raises were governed largely by the elevation of the bottom of the orebody. For high-lying orebodies raise development is used, haulage drifts are extended on 120-ft. centers and the raises are at 60-ft. intervals along the drifts. This raise-development method follows closely conventional block-caving practices with the draw fingers spaced at 15-ft. centers in both directions.

In the method of developing the lower-lying ores, the drift spacing is reduced from 120 to 60 ft., and the raises in the drift are at 30-ft. intervals, but the 15-ft. finger spacing is maintained. Drifts were never spaced closer than 60 ft.; therefore, where the orebody bottom approached the haulage level the branch raises pierced the ore, resulting in a part of the orebody being left below the undercut horizon. This ore, which was reclaimed later, ranged in thickness up to a maximum of 60 ft. from the haulage level to the undercut horizon. Scrapers and slusher hoists were first used in salvaging these unmined portions of the orebody, which, when broken, would not flow by gravity to the haulage drifts.

Evolution of Slusher-Drift Methods—At the Emma Nevada mine, to reclaim ore below the undercut level, pony sets were installed over the main drift caps at 30-ft. intervals along the drifts. Crosscuts, now known as slusher drifts, were then driven from these pony sets just beneath the ore. Where the orebody was 15 or 20 ft. above the haulage level, short raises were run from the pony sets to the ore and the slusher drifts driven at that horizon. The slusher drifts were timbered with 12x12-in. drift sets on 5-ft. centers. Short, cribbed fingers were installed at every third drift set, thus maintaining a 15-ft. finger spacing. The tops of these fingers were connected by undercutting. Where slusher drifts were driven from the pony sets at the same elevation, the ore was drawn through the fingers into the drifts and scraped directly into ore cars. Where a short raise extended from the pony set to the slusher drift, the ore was dragged into the raise; a chute with an arc gate was provided for loading the cars. The length of a slusher drift was limited

to 60 ft., as the main haulage laterals had been driven on that spacing. As portions of the orebody left below the branch-raise undercut horizon were mined successfully by this method, it therefore seemed logical, when the bottom of the orebody was close to the level, to dispense with the branch raises and attempt to mine the entire thickness through slusher drifts. Such a procedure in virgin area would save the cost of the branch raises and one undercutting operation.

In the latter part of 1937 it was decided to develop a trial block for slusher mining. An area was chosen where the bottom of the orebody was 15 or 20 ft. above the haulage level. The block developed lay between two haulage drifts 180 ft. apart; slusher drifts were driven on 30-ft. centers at the bottom of the ore between these drifts. Slusher hoists were installed at each end of the drifts, giving a slushing distance of 90 ft. Many difficulties were experienced in mining this trial block. The block was mined successfully, however, at a cost comparing favorably with the old practice, and a remarkably good recovery of ore was made. The possibility of applying this method of mining to other areas was next considered. A section along the north fringe of the orebody had been mined by using branch raises early in the life of the mine. This portion of the orebody had decreased in thickness and copper tenor to such an extent that the cost of mining through raises from the main level was prohibitive. A method of development was devised that utilized the raises that had already mined the overlying ore and that had since been abandoned. Slusher drifts were driven northerly at 30-ft. intervals from the abandoned raises. As the drifts advanced and fingers were installed, an occasional prospect raise was extended from the fingers through the ore. If enough ore was shown to justify development and mining, the drift was advanced. By this method the orebody was explored and developed in one step. This application of slusher mining has since been extremely useful in fringe areas, as the maximum amount of ore is recovered and the necessity of expensive churn drilling or other preliminary exploration is eliminated.

Continued exploration by churn drilling had disclosed that, in addition to the 7,000,000 tons recognized when the shaft was sunk and the haulage level established, there was a large tonnage of ore to the west. This westward extension of the orebody lies at a higher horizon than the main portion of the body and often is 200 ft. above the established haulage level. These higher orebodies were expensive to develop from the haulage level by the conventional branch raise methods. A comparison between the costs attained by slusher drifts in the fringes and low-lying areas and the branch-raise method indicated that the slusher method might be used to advantage in the high-lying ores. Studies indicated that the savings in development by applying slusher practice would more than offset any increase in cost incident to mechanical handling of the ore.

Two methods, known as standard slushing and sublevel slushing, were developed for extracting high-lying ores. Raises from the main haulageway have been placed 240 to 270 ft. apart, giving slushing distances of 120 to 135 ft. This distance has been exceeded in some instances, but present practice indicates that 120 to 150 ft. is the economic limit of slushing distance. The raises branch to give openings at or near the bottom of the ore on 30-ft. centers. The slusher drifts are driven from these raises just beneath the ore, and fingers are installed spaced 15 ft.

Sublevel slushing also has been applied in several areas. In this method the subslusher drifts are 60 ft. apart. Intermediate raises are run from the subslusher drift at 30-ft. intervals, and at the top of these raises a conventional control set is installed with four draw fingers. This method of development, as well as the standard method, results in a finger spacing of 15 ft. The main raise spacing along the haulageway may be increased somewhat over the spacing used in the standard slusher method, as the subslusher drifts are longer, are in better ground, and usually stand better, requiring less maintenance. It has been found that the slushing distance in the subdrifts can be as great as 150 ft.

Before a final decision is made on the mining method to be used in a block, a comparative analytical study is made of available data. Plate-glass models showing proposed development layouts, motor haulage drifts, and the ore-waste contacts are constructed. These models aid in solving the development problems, which otherwise would be difficult to visualize. In one particular block preliminary estimates showed a cost of 26.8c. per ton for development by standard raises, 18.7c. for development using subslushing methods, and 16.2c. for development by standard slushing methods. This block was located at a place where the orebody bottom was 200 ft. above the level and the ore averaged 125 ft. in thickness. When the orebody is closer to the level and of greater thickness the development savings are smaller, and in some instances conventional block-caving methods are used. Thus far conventional block-caving has been applied where the orebody is thick and 60 to 100 ft. above the haulage level. At the time of writing the mine is yielding about 70,000 tons per month by standard slushing, 80,000 tons by subslushing, and 30,000 tons by conventional block-caving methods.

Mine Openings—As virtually all of the ground in the Emma Nevada mine is extremely soft, heavy timbering is required to support most mine openings. Drifts and raises are kept as small in cross-section as is practical for handling large tonnages. Timbering in the main haulage drift and extraction laterals consists of a cap 6 ft. 6 in. in the clear, which is bridged and spiled in heavy or swelling ground; and posts that are 8 ft. 8 in. long on the dry side and 9 ft. on the ditch side. The posts generally are set with a batter of 1¾-in. per linear foot of post, which may be varied as ground conditions change. Twelve by twelve-inch Douglas fir timber is used for caps and posts and split lagging for top and side lacing. Ten-ton trolley locomotives and drop-bottom cars with a 5¼-ton pay-load capacity constitute the haulage equipment. The track is 75-lb. rail and has a gage of 36 in.

Raises—All raises are cribbed, the timber size being determined by the character of ground and the tonnage that is to pass through the raise. Where the ground stands fairly well, 4x12-in. cribbing generally is used, although 6x6-in. timbers may be installed when a raise is expected to accommodate a large tonnage and consequently will be subjected to excessive wear. In heavy ground, 6x12-in. or 6x8-in. cribbing on 6-in. spacing blocks is commonly used.

Where conventional block-caving is used, the chute compartment is 3½x3½ ft. on the inside; the manway compartments have inside dimensions of 2 ft. 2 in. by 3½ ft. In slusher mining a raise must serve about eight times the tonnage handled in conventional cave mining, and for this reason the manway and chute compartment are made the same size and are interchangeable. The compartments are 4 ft. square inside and lined on all sides with 2-in. lagging. Large tonnages pass through the slusher raises daily and prolonged delays for repairs would be costly. If the chute compartment has to be repaired, the manway can be converted quickly into a chute by taking out the ladders and installing a gate in the pony set. The manway compartment has ample room to accommodate the ladderway and a slide for hoisting timbers and other supplies.

Although hand-operated arc gates are used in the conventional mining raises, air-lift, down-cut, vertical gates have been installed for the slusher raises where much larger daily tonnages must be loaded. The air gates speed loading and reduce loading costs materially. One man on an air gate has loaded up to 2,000 tons in a single shift.

Excerpted from "Information Circular No. 7135, Kimberly Method of Block-Caving Using Slusher Drifts." By John A. Richards. November, 1940.

(*To be continued*)

Kimberly Methods of Block-Caving Using Slusher Drifts—II

IN THE standard slusher drift, sets are placed on 5-ft. centers except at the fingers, where a 4-ft. interval is used. The posts are plumb and are of 12x14-in. timber 6 ft. 2 in. long. Drift caps and sills are framed the same and are interchangeable; they are of 12x14-in. timber and 5 ft. inside the frame.

An important feature of this set is the 12x14 in. sill. Light sills failed as rapidly as they were put in, and the posts were driven down and pushed in at the bottom. Most failures occurred in former sets at the junction of the sill and post, which is a difficult place to relieve when the muck is pushed in between the posts during the slushing operation. The present drift set is designed to overcome this particular difficulty. Split lagging is used for tops and sides.

It is of paramount importance to provide a good drift bottom for slushing operations. The sill must be strong enough so that it will not buckle and break. The practice of installing three light steel rails as a scraping surface has proved very satisfactory.

Drift and finger crew consists of three miners, who work on a cooperative bonus system. Generally, two men work in the face and one man works on the fingers. The crew usually advances the face 5 ft. per shift, and the man on the fingers keeps pace with the drift crew. The miners place all of the timbers, including segment sets and a temporary 2-in. floor for scraping. The average time required to slush out a drift round is 40 min.

Mining Practices—The "peanut" porphyry immediately below the ore is quite soft and badly fractured. The ore-waste contact is nearly always marked by horizontal shearing and a gummy plastic material known locally as chewing-gum gouge. Layers of this gummy material may comprise 2 to 15 ft. of the rock immediately beneath the ore, and in some cases 10-ft. thicknesses of gouge have been found. In the first blocks mined by slusher methods, difficulty was experienced in maintaining the slusher drifts, particularly when this gouge was encountered. Much thought has been given to choosing the best locations for these drifts.

The success of slusher mining hinges almost entirely on the design, excavation, and maintenance of the slusher drifts and draw fingers. The best horizon for the drifts in the Emma Nevada mine is about 10 ft. below the ore-waste contact. As the bottom contour of the ore undulates, slusher drifts are laid out to strike this distance as an average. It has been found necessary to keep the drifts nearly horizontal; the maximum grade used is 4 in. per 5-ft. set. In grades steeper than this the drift sets ride upgrade and are difficult to maintain. Where the ore-bottom contour dips steeply or is step-faulted, ideal layouts are not possible. The ore may be 25 or 30 ft. above the drift at one end and may dip down into the drift at the other end. This irregularity is met satisfactorily in the finger detail and salvaging operations. Where the ore is high above the drift, the fingers are extended up to the ore contact. The excavated material is handled as waste where the drift or fingers are below the ore. The fingers have a minimum length of 10 ft. above the cap, 5 ft. of which is timbered. Any cones of ore left between the fingers are recovered as a salvage operation after the fingers have been drawn to completion.

When a raise for standard slushing has been completed and the cribbing leveled off at a point generally about 18 ft. below the ore, a chute and manway set are installed on top of the raise. The tugger station next is excavated and timbered, and after two or three drift sets are placed, the tugger hoist is set up to scrape the spoil made in drifting into the raise.

To minimize maintenance it has been found essential in slusher-drift and finger excavation to keep overbreaking at a minimum, and the development crews are penalized if they overbreak where it is avoidable. By careful development the pillars between the slusher drifts and the cones between the draw points are kept to a maximum size, which aids materially in supporting the overlying weight and in reducing repair costs.

The draw fingers are placed on 14-ft. centers along the slusher drift and constructed of 8x8-in. cribbing resting on 10x10-in. bearers. The fingers are 3x3 ft. inside and installed at a 55-deg. angle from the horizontal. The timbered portion of the finger set extends 4 ft. above the drift sets. If the fingers are in ore, one vertical round is taken out above the cribbing and is belled out on the footwall side. If the fingers are in waste, they are extended up to the ore without cribbing and the last round is belled out on the footwall side.

The finger is designed so the chute blaster can punch the ore down or blast the finger from the drift. Long cribbed fingers were tried in the early stages of slusher mining, but the cribbing was crushed or blasted out and maintenance was excessive. It was also found to be too hazardous to climb into long fingers in the event of hang-ups.

The length of the slusher drifts has a direct bearing on the economy and success of the method, as savings in main level and raise headings are increased as the slusher drifts are lengthened. It has been found that about 120 ft. is the most economical length, and 150 ft. is the maximum. Where ground conditions are bad and much repair work is necessary, it is impractical to have drifts longer than 120 ft., as the collapse of a part of the drift takes too great an area out of production while repairs are made. All the slushing is done on one shift at the Emma Nevada mine, leaving the other shift free for repairs. The amount of ore that can be slushed in one shift through a 120-ft. drift is used as a control figure for mining in that area. Length of the slusher drifts may be increased for very thin orebodies, as the drifts will have a shorter life and heavy repairs may not become necessary.

Undercutting—The method of undercutting is designed to keep as much solid ground as possible between the top of the draw point and drifts to protect the drifts from excessive weight. All undercutting is done with self-rotating stopers, and all holes are drilled at 45 deg. or steeper. This results in pyramids of solid material being left between all draw points. Careful supervision is necessary so that the undercut may be complete, with no pillars left. These pillars would throw undue weight on the drifts and fingers and cause piping when the ore was drawn. It is considered as important not to overbreak into adjacent virgin ground as it is to underbreak and leave pillars. Drawings are furnished the undercut bosses and the instructions are rigidly enforced. The weakest point in the orebody is near the contact, and undercutting must be rapid to prevent caving, which might draw out the crew before the undercut is complete, so that pillars are left.

The undercutting is done on a bonus system, and a crew consists of two men. An average of about two shifts

is required to complete a unit, which comprises one pair of fingers, or 450 sq.ft.

Undercutting is begun above the back end of the drift and retreats to the main rise, always breaking to the cave. The first stage in undercutting, where the finger is in ore, is to drill two raise rounds and bell out the last round. If the finger is in waste, it is extended up to the ore and the last round is belled out. The second stage is to connect the fingers by inclined connection drifts that parallel the slusher drift. This connection is made with as large a cross-section as the ground will permit to take care of the swell when the final round is blasted. The last stage in mining is to drill out the back and pillars from the inclined connection drift, using eight radial rounds, 4 ft. apart, of five holes each. It has been found impractical and unsafe to partly undercut large areas and shoot out a large group of pillars at once.

As the ore is very soft and thoroughly cross-fractured, cut-off stopes are necessary only in exceptional instances. In the harder parts of the orebody or in blocks bounded on all sides by virgin ground, very short shrinkage cut-offs are used in the corners of the block.

When the undercutting is completed, a bottom slice of ore 30 ft. wide, 8 ft. or more high, and the length of the slusher drift has been removed.

Slushing Detail—As soon as the undercut is blasted miners begin drawing ore. The newly undercut fingers often have pockets of gas trapped behind the ore in the fingers. Because of the danger of one man being overcome by gas and not being rescued immediately, two men work together. Much hand augering and blasting are necessary to induce a recently undercut area to cave, and this also requires two men. At this period in the life of a slusher drift, both shifts are occupied in drawing the broken ore. Within a few days the newly undercut block starts caving, and normal operations are begun.

When a block has finally been induced to cave by drawing the blasted material out of the undercut portion, all drawing and slushing in each slusher drift are done on one shift by two miners. The repair and maintenance work is performed on the opposite shift. Two men constitute a repair crew, and the mine safety rules require them to work within sight and sound of each other. The number of repair crews necessary depends on the amount of work to be done in each place. Under ordinary conditions three repairmen, or one and a half crews, handle all the mining and repair work for any slusher drift.

As pulling ore and repairing are specialized jobs requiring a practiced routine and a developed skill, an attempt is made to keep the trained men on their specialized jobs. As a general rule the younger, more agile men become better slusher operators, while the older and more experienced miners do the maintenance work.

Interesting experiments planned for the near future include a trial block using a 30-hp. hoist for cross slushing to effect further development savings and installation of a shaking conveyor to replace the hoists and scrapers in the slusher drifts.

Draw Control—The success of a caving method of mining depends directly on an efficient practice of drawing the ore; the maximum recovery of ore with minimum dilution of waste is sought. Sound knowledge of the mechanics of caving combined with experience and close observation and systematically recorded data are the prerequisites for success. At the Emma Nevada mine every means is taken to insure the proper draw of ore and to record the data so that they can be interpreted readily.

Advantage of Plate-Glass Model

One of the best aids in the drawing procedure is a plate-glass model, which is made for each block. Various types of models, charts, and graphs were tested before the glass model was tried. The plates represent 30-ft. vertical sections, and each block has four to eight glass sections. The plates are held at the base by grooved blocks. The scale used on the sections is 20 ft. to the inch.

The development openings are shown on the bottom portion of the sections and the top and bottom contours of the ore depicted above. The ore is shown in solid red by spraying the color on the glass plates. Development workings are in black. At ten-day periods a portion of the red color corresponding to the height of ore that has been drawn out is scraped from the plates. These plate models have an advantage over other models in that the relative positions of the draw points are in full view. Of course these models are valueless unless they are kept strictly up to date. The drawing procedure is under direct supervision of a draw-control boss, who has charge of and is assisted by the samplers. The draw boss, samplers, mucker bosses, and foreman all work in close harmony.

At the beginning of a shift each slusher operator is given a draw slip, which is a printed small-scale drawing of the slusher drift. The number of cars to be drawn is marked on each finger, and any special instructions are written on the slip. At the end of the shift the slusher operator fills in the number of cars actually drawn from each finger. The slusher operator keeps track of the number of scraper loads from each finger and a factor is used to convert number of loads into cars.

The fingers are constructed in such a way that the broken ore is under control at all times. Each time the slusher operator punches down a finger, about three scraper loads or 2.1 tons of ore run into the drift. Over a period of time all the fingers will average about three scraper loads for each time they are drawn. The slusher operator always begins drawing the fingers from the back end of the drift. Because the full length of the drift is visible, the supervisors immediately investigate if any finger is not being drawn. With the slusher method of mining, control of the draw is ideal. This conclusion has been borne out by the excellent recovery of ore effected with a minimum amount of dilution.

An average slusher-mining drift has 14 fingers; with a draw of 20 tons per finger, the daily production for a drift is 280 tons. This tonnage is mined each day in a series of cycles consisting of a slushing period and a finger-pulling operation.

The slushing operation lasts 15 to 35 min., during which time the slusher will handle approximately 1 ton of ore per minute. Each trip of the scraper moves 0.7 ton of ore an average distance of 75 ft., requiring 40 sec. to complete the round trip, including loading and dumping. The slusher operator soon becomes expert. An average of about 2 sec. is required to load the scraper. The load is dumped with virtually no more time lost than is required in the process of reversing the direction of the scraper.

The finger-punching operation, which begins when the drift is slushed clean of ore, lasts 10 to 20 min. The ore packs easily, and a finger seldom runs for any length of time without punching. The miner starts on the last pair of fingers in the slusher drift and bars down the ore into the drift, working the fingers back toward the tugger. His main tool is a bar 5 to 7 ft. long made of 1-in. pipe forged to a point at one end. A finger may hang up for one of two reasons—either it becomes packed or the coarser ore becomes tightly wedged. The packed condition is alleviated by drilling hand-auger holes and blasting. The wedged boulders are blasted by bombs made of a number of sticks of dynamite tied to the end of long, light, blasting sticks and placed tightly between the boulders. The finger-drawing and slushing cycles are continued each shift until lunchtime unless hung-up fingers must be blasted earlier. Plugged fingers generally are blasted at lunchtime.

(*To be continued*)

Excerpted from Information Circular No. 7135, Kimberly Method of Block-Caving Using Slusher Drifts. By John A. Richards. November, 1940.

Kimberly Methods of Block-Caving Using Slusher Drifts—III

ROUTINE slusher-drift repair work consists of changing top and side lagging and removing ground where weight begins to show on the timbers. Light repairing is the most important work of the repairmen and if promptly taken care of heavy repairs may be avoided. To keep a slusher drift in condition so that no delays shall hinder the ore-pulling shift, all the broken top lagging must be changed daily. An experienced miner changes a broken top lagging in a few minutes; consequently, a crew can readily finish this part of its work in a half shift. The broken side lagging is next replaced and set back to keep the ground from the posts. The light repairing and finger blasting are routine, and no special orders are given for their accomplishment. Finger blasting is the last duty of a repair crew and occupies the final 45 min. of their shift. The packed fingers are drilled by hand auger. Large boulders are drilled with a hand-rotated stoper, and fingers containing smaller boulders are bombed. All shots are blasted going off shift.

Changing of broken posts, caps, or girts is termed heavy repair. If light repairs are kept up properly, heavy repairs are seldom necessary. Heavy repair, because of the danger of closing a slusher drift, is given preference. If a post, cap, or girt breaks, the boss orders it changed immediately, and the squeezing ground or boulder that broke the timber is removed.

In slusher mining as practiced at the Emma Nevada mine it is essential that drifts be kept open for the daily draw. Experience has shown that a steady, uniform flow of ore from all draw points is necessary to prevent development of excessive weight on the extraction openings. If a drift is closed for heavy repairs at any point over a period of a few days, the drift timbers beyond that point invariably crush. The method used in maintaining the slusher drifts is most economical, as a small amount of light repairs at the proper time and place saves a great deal of heavy repair work later on.

Slushing Equipment—The equipment best suited for slushing is an individual problem, depending on the character of ore and general slushing arrangements. Sixty percent of the ore mined at the Emma Nevada mine will pass a 1-in. screen and about 20 percent will pass 65 mesh. Large boulders are extremely rare, and a slusher drift often produces its daily quota of 300 tons with only eight or ten boulders showing that require breaking to pass the standard 11-in. grizzlies. On the other hand, the ore is sticky and abrasive.

Although cast scrapers of various makes have been tried, it has been found that a hoe-type scraper designed and fabricated in the company shops is the cheapest and best for Emma Nevada ores. It is recognized that a heavier, sturdier type would be required for coarser material and under more adverse slushing conditions.

Ropes—Many different types of wire ropes have been used, and performance data have been recorded carefully. The best results have been obtained with ½-in. 6 by 19 preformed improved plow-steel ropes with hemp center. Broken ropes are spliced by men designated for this work, who are responsible for the general care and lubrication of the slushing equipment. Ropes are knotted only as a temporary measure, as a knotted rope piles up on a hoist drum, snags in the blocks, and hastens wear and breakage.

Hoists — The slusher hoists are double-drum electric tuggers of standard manufacture in 15- and 20-hp. sizes. It is felt that compressed-air-driven hoists are not practicable for the large daily tonnages. Rope pull is 2,200 to 2,500 lb. Loaded speed is 200 f.p.m. on the 15-hp. units and 280 to 300 f.p.m. on the 20-hp. units. The 20-hp. units are used chiefly in the subslusher drifts, which serve an area 60 ft. wide and produce twice the daily tonnage of a standard slusher drift. The 20-hp. units handle 48-in. scrapers, while 42-in. scrapers are used with the 15-hp. units. Ten-inch head blocks and 4-in. and 6-in. guide blocks are standard for the haul-back line.

General — Slushing routine at the Emma Nevada mine offers no information as to what daily tonnages the slusher hoists could handle if continuous slushing were practiced. The actual operation is intermittent finger-drawing and slushing, and the tonnage to be drawn is governed by the rate at which the ore caves. The 15-hp. units will handle 250 to 300 tons over an average slushing distance of 75 ft. with an actual slushing time of four to five hours. The 20-hp. units have handled 500 to 600 tons over an average distance of 75 ft. in about 6½ hours actual slushing time. These daily tonnages represent the rates dictated by proper draw control. The hoists and scrapers are not unduly large and operate satisfactorily in the slusher drifts, which are kept small in cross section to minimize repair costs.

Advantages of Slusher Mining—The chief advantages of slusher and subslusher methods of mining compared to conventional block-caving methods are represented by the savings in development expenditures. The development savings have ranged from 3c. per ton to a high of 10.6c. per ton. Maximum savings are made when the orebody is thin and high above the haulage level. These development savings do not include the material savings also made in main haulage drifts when slusher methods are used.

The total footage of drifting saved by application of slusher methods is 7,000 ft., which would cost from $30 to $50 per foot, depending on ground conditions. With the decreased footage of haulage drifts and other development headings it follows that maintenance costs per ton will be reduced. The advantages in prospecting and mining fringe areas and the consequent beneficiation of ores that might otherwise be lost to mining have already been noted.

An important operating advantage of slusher mining is the ease of ingress and egress afforded by the development layout. Instead of climbing up and down ladders, the mine bosses may now examine entire sections by merely walking through the slusher drifts. The 16 draw fingers and a slusher drift 120 ft. long may all be examined in a much shorter time than would be required to climb into four individual square sets, as was necessary under the older practice. As a result, a boss can examine four or five times as great an area as heretofore, with much less physical exertion.

Draw control, the all-important factor, has been improved through better supervision. The samplers, control engineers, and bosses now see virtually all producing draw points daily, an almost impossible task when conventional block-caving methods are employed.

The slushing method insures a controlled draw, as fingers cannot "run wild." A chute puncher in a square-

set can fill his raise from one finger that is running well, but in slusher mining, when all fingers have been drawn into the drift, he must slush the ore to the raise and then repeat the process.

The ease of ingress and egress afforded by the development layout also aids better ventilation and makes delivery of timber and other supplies much easier and less expensive. Advantages in repair and maintenance are equally important. It is much easier to carry on repair and maintenance work in a drift than on the top of a 100-ft. raise.

Incidentally, in one area in which the ores were particularly high-lying it was found advisable to drive a vertical raise from the main haulageway to the mining horizon to serve the slushing area as a supply and manway raise. The raise was equipped with an Otis elevator having a live-load capacity of 5,000 lb. Connection drifts were driven from this raise to the various slushing drifts so as to serve the area with supplies and provide easy entrance for the workmen. The openings also provide better ventilation. This installation paid for itself before one-third of the ores in the area served had been mined.

In slusher mining the number of raises from the main level to the mining horizon is about one-eighth those required in conventional block-caving. A smaller number of loaders and "chute knockers" is required for a given tonnage, and haulage problems are simplified, resulting in an appreciable saving in loading and haulage.

By using slusher mining methods, working conditions have been made safer in many respects. It is axiomatic that improvements in supervision and ventilation go hand in hand with safety. Horizontal mine openings with easily accessible connections are safer than raises, and the miners can aid fellow workmen much more quickly and with greater ease. Perhaps the greatest contribution slusher mining makes to safety is the fact that workmen are in closer contact with each other.

Costs—Since the inception of slusher mining, separate costs have been kept, where feasible, for conventional block-caving, slusher mining, and subslusher mining. A complete segregation of costs, however, has been wholly impracticable. Direct labor and supply costs are segregated, but loading and tramming charges are merely divided on a tonnage basis, resulting in the same recorded costs for the three methods. Total loading and tramming charges are between 8 and 9c. for the entire mine, and time studies indicate that the amount of these charges properly attributable to slusher mining ranges from 6½ to 7½c. Improvements in ventilation, supervision, and delivery of timber when slusher methods are used are not reflected in the costs as recorded.

The men who operate the tuggers have many other duties common to both methods of mining. They draw the fingers, bomb and drill when necessary, and help keep the slusher drift and fingers in repair. It is impracticable to record the actual hours and minutes spent at each particular task, and for that reason the cost sheets do not reveal the exact cost of mechanically handling the ore. The total of such expense however, is included in the cost of "caving," which is between 11 and 12c. per ton for all charges incident to drawing the finger and slushing the ore to the raises. Studies indicate that the cost of actual slushing, including labor, power, cables and blocks, scrapers, maintenance of all equipment, and amortization of the hoists on a two-year basis, does not exceed 4c. per ton.

The accompanying table shows comparative costs for the first nine months of 1939. Cost of raises, control drifts and fingers, and undercutting does not show, as the amount expended depends merely on the amount of this work undertaken in a given period. The cost per ton for these items is also governed by the thickness of ore in the blocks being mined.

Costs for First Nine Months of 1939

	Raise Mining	Slusher Mining	Subslusher Mining	Totals
Tons mined	335,051	587,443	692,407	1,614,901
Caving expense	$0.068	$0.109	$0.117	$0.105
Repairs	0.068	0.072	0.084	0.076
Loading	0.019	0.019	0.019	0.019
Tramming	0.063	0.063	0.063	0.063
Hoisting	0.033	0.033	0.033	0.033
General mine expense	0.109	0.109	0.109	0.109
General expense and maintenance	0.101	0.101	0.101	0.101
Totals	0.461	0.506	0.526	0.506

Past records and estimates for future mining show a charge of 15c. per ton for all development openings above the haulage level and about 6c. for main-level development for the entire mine, which when added to the 50.6c. gives a total cost per ton of 71.6c. This figure includes all items of corporate expense except capital charges for construction items and major equipment and income and bullion taxes. General expense items are high, as the ores are concentrated and smelted on a custom basis, leaving the mine only to bear general charges.

A true answer to the economy of slushing practice cannot be gained without studying the development layout for each individual block. A block mined by slusher methods may show a cost per ton higher than a neighboring block mined by raises, because the ores were thin and high above the level, with resulting high development charge; yet material savings may have been effected.

In analyzing the cost tabulation shown above it must be remembered that the standard raise-mining methods were carried on in areas where the orebody bottom was at an ideal distance above the haulage level (approximately 60 ft.) and where the ore was of greater than average thickness. Apart from the obvious effect of lower development costs per ton, slusher mining will lead in many other ways to cheaper mining costs. Attention is called to the fact that slusher mining has been used in places where the less favorable conditions exist, such as thinner fringe areas and areas where the ore is high above the level.

Excerpted from Information Circular No. 7135 U.S.B.M., Kimberley Method of Block-Caving Using Slusher Drifts. By John A. Richards. November, 1940.

Differential Grinding of Alabama Iron Ores For Gravity Concentration

RECENT WORK on beneficiation of the Alabama red iron ores and ferruginous sandstones has brought out interesting new facts on particle-size reduction that are a valuable supplement to what is already known. The work was done by the Southern Experiment Station of the Bureau of Mines in cooperation with the University of Alabama, University, Ala., on two samples from the Birmingham district. One was an oolitic red iron ore containing about 35 percent iron; the other was a ferruginous sandstone containing about 25 percent iron, particularly attractive because of its low phosphorus content. In both samples rounded quartz (sand) grains were embedded in an oolitic, "onion-skin" matrix of hematite. The hematite carries varying amounts of clay, which limits the grade of concentrate that can be made. The clayey hematite is soft, so that the rock crumbles readily under pressure. The sand grains shell out like corn, and without much exaggeration the mill used for differential grinding might be called a "sheller."

Nature of Sand Grains

The outstanding fact is that the particle size of the sand in the two samples is different. Table I shows that the predominating size of the sand in the red ore is 35-mesh, whereas the predominating size in sandstone is 100-mesh. About half (46.7 percent) of the ferruginous sandstone is rounded quartz sand.

In the sizing tests of the sand the ores in lump form were treated with hydrochloric acid to dissolve the hematite, the solution was removed, and the residue was screened in neutral water, a rubber muller being used to disintegrate the particles of clay. A fine, clean sand was thus obtained as a final residue. Examination of the shape characteristics of the particles in these residues, size by size, under a comparator microscope indicates that the sands in the two ores may not have the same origin, but this question has not been answered. However, all grains were predominantly rounded—an important characteristic in beneficiation by gravity methods. If they were angular and flaky after the manner of crushed quartz, satisfactory gravity concentration of these ores would be more difficult; in fact, it would be impossible. Being rounded, they roll and race freely to the side of the table in table concentrations and settle freely in a bowl classifier, where a slime of concentrate grade is removed over the top of the bowl.

The hematite that is cemented to the sand grains and persists as an iron stain is beyond recovery by any known physical separation. Hence, such a loss would appear to be proportional to the amount of surface of the sand. Assuming that this theory is correct, a unit weight of the ferruginous sandstone would have about five times as much sand-hematite interface as an equal amount of red iron ore (Table I). If the assumption is carried further to consider percentage recovery, it may be shown that for each unit of iron in the ferruginous sandstone there is seven times as much sand surface as in the red ore. On this basis the red ore would be expected to give a higher percentage recovery. However, this assumption does not hold true by tests—the ferruginous sandstone yields as good a recovery as the red iron ore. Hence, the conclusion is reached that the cementation is not so severe in the ferruginous sandstones. They yield a lower-grade tailing in spite of the preponderance of sand surface; the "shelling" of the ferruginous sandstone releases cleaner quartz grains than the red iron ore.

The sand grains have the customary hardness of quartz, but the cementing hematite is relatively soft. Hence, there is a difference in resistance to particle-size reduction, and this difference must be utilized if gravity concentration is to be applied. Differential grinding—that is, reduction of the soft particles while the hard particles are left in their natural discrete state—was therefore practiced. This unique method of grinding is the essence of the success that has been achieved in developing a simple flowsheet for gravity concentration. The remarkable part about the differential grind is that the hematite is greatly reduced in size and liberated, while the sand is not attacked. In fact, from a practical standpoint the hematite approaches a colloidal state. On the other hand the sand has a limit of fineness. Table I shows that the sand finer than 200 mesh may be regarded as absent. Hence, when differential grinding is properly applied much of the hematite may be taken off by desliming methods common in milling.

If all the hematite yielded a slime in differential grinding, the recovery could be made to approach closely to 100 percent, but its nature does not permit such an ideal recovery. Part of the hematite is hard, highgrade, and resistant. and for maximum recoveries gravity concentration is necessary. Furthermore, in the red ores oolites high in lime induce a loss by their resistance to treatment. A red ore with a low lime content is expected to be most benefited by the differential grinding.

The red iron ore may be reduced to ¾ in. by any convenient method. When reduced to this size about equal amounts are retained on or pass through an 8-mesh screen; furthermore, the iron content is not segregated appreciably in the two sizes. It is segregated only at sizes finer

Table I—Screen Test on Sand From Iron Ores

Mesh	Red Iron Ore, Weight, Percent	Ferruginous Sandstone, Weight, Percent
Minus 10 plus 14	1.1	
Minus 14 plus 20	7.7	Trace
Minus 20 plus 28	17.3	Trace
Minus 28 plus 35	23.5	0.1
Minus 35 plus 48	21.5	1.3
Minus 48 plus 65	12.5	23.9
Minus 65 plus 100	9.6	50.1
Minus 100 plus 150	4.7	16.7
Minus 150 plus 200	2.1	3.4
Minus 200	...	4.5
	100.0	100.0
Sands in ore, percent	20.3	46.7
Fe in ore, percent	35.9	28.5
C_aO in ore, percent	10.2	0.06

than that of the predominating amount of sand.

Further reduction of the ore must be made by a well-designed method that will not crush the quartz particles. However, since concentration at 8 mesh is anticipated, one-half has already gone through the first step in preparation by the reduction through ⅜ in. This may be termed the "natural" minus 8-mesh size to distinguish it from the coarser portion. Some of the characteristics of this material are shown in Table II.

Table II—"Natural" Minus 8-Mesh Product From the ⅜-in. Grind

Mesh	Weight, Percent	Assay, Percent Fe		Distribution, Percent Fe	
Minus 8 plus 100	77.2	32.1		70.5	
Minus 100 plus 150	3.5	40.3	45.2	4.0	29.5
Minus 150 plus 200	2.2	42.0		2.6	
Minus 200	17.1	47.0		22.9	
	100.0	35.0		100.0	

Table III—Reduction of the Red Iron Ores, Minus ⅜-in. Plus 8-Mesh, to 8-Mesh by Rolls and by Rod Mill

Mesh	Rolls: Weight, Percent	Rolls: Assay, Percent Fe		Rolls: Distribution, Per cent Fe		Differential Rod Milling: Weight, Percent	Differential Rod Milling: Assay, Percent Fe		Differential Rod Milling: Distribution, Percent Fe	
Minus 8 plus 100	86.0	34.6		82.5		73.3	32.3		65.0	
Minus 100 plus 150	2.8	42.6	45.0	3.3	17.3	2.6	41.3	47.9	3.3	35.0
Minus 150 plus 200	1.7	43.3		1.9		2.3	42.6		2.7	
Minus 200	9.5	46.2		12.2		21.5	49.3		29.0	
	100.0	36.0		100.0		100.0	36.5		100.0	

Table IV—Sizing Analysis of Rod-Mill Discharge, Ferruginous Sandstone

Size	Weight, Percent	Size	Weight, Percent
0.371-inch plus 3-mesh	8.1	Minus 28- plus 35-mesh	0.9
Minus 3- plus 4-mesh	17.8	Minus 35- plus 48-mesh	0.9
Minus 4- plus 6-mesh	15.4	Minus 48- plus 65-mesh	4.5
Minus 6- plus 8-mesh	11.4	Minus 65- plus 100-mesh	8.9
Minus 8- plus 10-mesh	6.1	Minus 100- plus 150-mesh	4.2
Minus 10- plus 14-mesh	4.5	Minus 150- plus 200-mesh	1.5
Minus 14- plus 20-mesh	2.3	Minus 200- plus mesh	12.5
Minus 20- plus 28-mesh	1.0		100.0

The table shows that a severe desliming may be expected to take off 29.5 percent of the iron in a slime with a tenor of 45.2 percent iron, but if the desliming is more precise and at 200-mesh, a tenor of 47 percent iron may be expected. The table concentration of the coarser sizes has been described in several Bureau publications.

Means of reducing the ⅜-in. to 8-mesh size will now be considered. Only two methods will be mentioned here—roll grinding and differential rod milling. The results of laboratory tests are shown in Table III.

Although the roll grinding was performed in closed circuit and with great care to avoid reducing the sand particles, it hardly seems acceptable when compared with the rod milling, which yielded 35 percent of the iron in a slime with the tenor of 47.9 percent iron. Hence, only the rod milling will be discussed.

The rod milling was done in a laboratory mill 19 in. in diameter and 36 in. long, with rods 1½ in. in diameter; the discharge opening was 8 in. in diameter. The practice was a special type, not the ordinary rod milling with a full load of rods, nor was it "self-grinding" with the grinding medium omitted. A full load of rods would have minimized differential grinding, and self-grinding would have yielded too small an output. Hence, a rod volume of only 13 percent (300 lb.), instead of the customary 45 percent, was selected.

Ordinarily a rod mill is half full of rods, leaving space for ore only between the interstices of the rods and in the pool, but here the small rod volume leaves space for a great amount of ore, some of the particles of which will be subjected to attrition by tumbling upon each other. The speed of the mill must not be high enough to cataract the rods against the breast of the mill, as such cataracting would crush some of the quartz grains.

The "natural" fines shown in Table II, although already through 8-mesh, needed some differential grinding to remove more of the hematite veneer from the quartz grains. Consequently the minus ⅜-in. product would not be screened on 8 mesh, as was done to dissect the problem, but would be sent in its entirety to the rod mill. Thus screening to ⅜-in. size at the head of the rod mill completes the preparation of the feed.

Differential Grinding of the Ferruginous Sandstone

Except for the size of the screens at the discharge of the mill, the flowsheets for the two ores will be identical; both ores will be reduced to ⅜ in. for the feed to the rod mill, and the middlings from the tables will be returned to the rod mill for further treatment. The only difference lies in the mesh of the screens at the end of the rod mill. As already stated, and as would have been predicted by examination of Table I, the red iron ore requires an 8-mesh screen. Similarly, the ferruginous sandstone requires a 30- by 30-mesh screen. The finest screens permissible should be used to reduce the last clusters of fine sand still cemented by hematite. Yet the screens must be coarse enough to permit the coarsest sand grains to pass. Table I is a guide in this matter.

Doubtless the first impulse of the operator will be one of alarm about screening an iron ore through 30 mesh, and hence it must be explained that the product is unlike the discharge of ordinary grinding machines. Ordinarily the critical size is in large amounts, but here it is very small, as shown in Table IV, a sizing analysis of the mill discharge. The 20- to 28-mesh size is only 1.0 percent of the whole. Furthermore, the sharp and angular grains presented in ordinary ball milling are absent.

The feed was through ⅜ in., and the circulating load was 250 percent. The small amount at the critical size (30-mesh) makes the screening easy. It is believed that the customary classifier cannot compete where the screening is so simple. Each ore yields a concentrate approaching a grade of 50 percent iron, and the recoveries will be between 80 and 90 percent. But the use of a lightly loaded rod mill run at a moderate speed to differentially grind a feed as coarse as ⅜ in. is a marked improvement over the flowsheets formerly recommended.

Excerpted from U. S. Bureau of Mines Reports of Investigations 3523, by Will H. Coghill, supervising engineer Southern Experiment Station, Bureau of Mines, Tuscaloosa, Ala., and Philip H. Delano, assistant chemical engineer at the same government station.

Methods and Costs of Mining Zinc-Lead Ores at Pend Oreille Mines & Metals Co., Metaline Falls, Wash.—I

THIS PAPER, which describes the mining practices of the Pend Oreille Mines & Metals Co. at Metaline Falls, Wash., is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in various mining districts of the United States. Acknowledgment is made to L. P. Larson, president, and Jens Jensen, secretary, of the Pend Oreille Mines & Metals Co. for permission to publish. Information was supplied by E. Berglund, mine superintendent; Earl Miller, chief engineer; Lynn Kinney, John Currie, and Wally Frick, engineers; Arthur Newman, chief clerk; and Jack Crampton, assayer.

Pend Oreille Mines & Metals Co. operates in the Metaline mining district of Pend Oreille County, northeastern Washington. The company properties are on both sides of the Pend Oreille River, north of the town of Metaline Falls. Metaline Falls is approximately 110 miles north of Spokane, Wash., and 11 miles south of the Canadian border.

History—Lead deposits in the Metaline district have been known for the past 60 years. In 1904 L. P. Larson started prospecting there, and in 1910 he interested the Lehigh Portland Cement Co., of Allentown, Pa., in a limestone and shale deposit, which prompted that company to erect a large cement plant at Metaline Falls. Mr. Larson gave the townsite of Metaline Falls to the State of Washington in the same year. Prospecting for zinc-lead ore was carried on during this period; a mill employing table concentration was erected in 1917 and was operated for a few months. For the next ten years there was little activity, but in 1927 the old Pend Oreille mill was rebuilt as a flotation plant and operated for six months on surface ores. Pend Oreille Mines & Metals Co. began development in the area in 1928 and discovered promising orebodies. The mill operated intermittently from 1930 to 1934. Development continued, and in 1935 construction of the present plant was begun. It has been operating continuously since Jan. 1, 1936.

Geology—The replacement of orebodies of the Metaline zinc-lead district in northeastern Washington are limited to the greatly disturbed fault block through which the Pend Oreille River flows and are associated with the major faults but are not in them. They are mostly near the top of the Metaline limestone, of Middle Cambrian age, and below black Ordovician slate. The ores are generally in jasperoid and are surrounded by a shell of crystalline dolomite. Outside of the dolomite is a shell of coarse-grained calcite that in places was deposited along solution channels long distances from known orebodies. Jasperoid, crystalline dolomite, and, to a less extent, coarse-grained calcite are therefore favorable but not infallible signs of the presence of ore near by. The Metaline limestone was dolomitized regionally before the ore was introduced, and this early dolomitization is independent of the ore.

In the area of Pend Oreille Mines & Metals Co. operations the ore bed lies 200 to 400 ft. below the surface. Gravel and sand overburden usually ranges from 10 to 150 ft., but in some places it is 250 ft. deep. The shale formation over the ore horizon ranges from nothing to 200 ft. in thickness. The favorable lime horizon is approximately 200 ft. thick, and the commercial ore occurs on or very near the footwall of the favorable lime bed in the Metaline limestones. The ore is in large, irregular, adjoining blocks close to the bottom of the favorable limestone beds. In most places the footwall is a massive gray limestone underlying the limestone beds. The general dip of the ore beds is 20 deg. to the north and east.

Physical Characteristics of Ore and Enclosing Rocks—The limestone beds in which the orebodies occur are very siliceous, brecciated, and faulted. The explored area is approximately 6,000 ft. along the dip and 3,000 ft. along the strike. Known commercial orebodies in this area have an approximate length of 5,000 ft. and a maximum width of 700 ft. The length, width, thickness, and dip of the ore horizon usually are controlled by faults the displacements of which are from a few inches to several hundred feet. Commercial ore ranges in thickness from a few feet up to 100 ft.; this change in thickness may take place within a very few feet horizontally, usually as a result of faulting.

The ore horizon, although badly faulted and fractured, is very solid. The walls and backs of mined stopes stand with very little sloughing. Stopes as much as 300 ft. in length, 100 ft. in width, and 120 ft. in height have been opened.

Large blocks of calcite occur in the ore horizon and usually are completely surrounded with ore. These calcite blocks are from a few inches to 200 ft. wide. Large blocks of low-grade ores (that is, up to 2 percent zinc) occur in the ore horizon. These low-grade sections are similar in size to the calcite blocks. The calcite and low-grade sections are used wherever possible for pillars and wall supports.

In places in the ore horizon are large open caves ranging from a few feet to 120 ft. in length, 110 ft. in width, and 15 ft. in height. The backs of the caves usually are fairly sound, and are lined with asbestos known locally as "mountain leather." The floors of the caves are covered with large pieces of waste, under which are high-grade zinc and lead ores. Only one cave has been opened that has not produced high-grade ores. The caves do not follow a definite line or formation and do not always occur on faults.

The ore is very spotty or irregular, even in the high-grade sections. Within 10 ft. a stope may range in grade from 30 percent zinc and lead to 2 percent zinc and lead. Ore as mined usually assays 5 to 7 percent zinc and 0.5 to 2 percent lead. The zinc concentrate averages 62 to 63 percent zinc with 0.2 to 0.3 percent lead, and the lead concentrate contains 80 to 84 percent lead with 1 to 2 percent zinc.

Methods of Prospecting and Exploration—In the Metaline district are large outcrops of the favorable limestone, many of which are mineralized. For years these outcrops received a great deal of attention. Mining was attempted on a few of them, but it was soon realized that the surface showings were very shallow.

As the district is very rough and steep and generally covered with overburden ranging in depth to 250 ft., the only method of prospecting that has proved satisfactory is diamond drilling from the surface. Diamond-drill holes are put down vertically, chiefly because it is easier to drive the standpipe through the great depths of gravel and sand. This practice has proved satisfactory also in getting representative cross-sections of the underlying limestone horizons.

Sullivan R-21 and R-22 diamond drills powered by 15-hp. gasoline motors are used on the surface. The R-21 drill has been in operation for five years and except for annual overhauling of the gas engine, maintenance cost has been negligible. The drills are equipped with hydraulic feeds. Extra-heavy, 3-in. standpipe is driven to bedrock; a 2¼-in. casing is then drilled 10 ft. into the bedrock; this is followed with "EX" casing, which is drilled 10 ft. beyond the bottom of the 2¼-in. casing. "E" rods are used for drilling and are pulled in 20-ft. lengths. Shale is drilled with a bull-nose bit that takes a core about ¾ in. in diameter. With this bit runs of 20 ft. are made between pulls. The footage per 8-hr. shift in shale averages approximately 50 ft. In the limestone, cast-set "E" bits are used with a double-tube core barrel. Runs are from 1 ft. to 5 ft. in length. The ground is very blocky, and to save core the rods must be pulled whenever the tube blocks.

Drilling is very slow through the ore horizon because of the hardness of the siliceous low-grade ore and the broken character of the ground. Runs usually are from 1 to 3 ft. in the ore and 5 ft. in the footwall limestone. The holes are stopped when they have entered 25 ft. into the gray limestone footwall. Sullivan Baby Turbinairs with screw feeds and a swivel or double-tube core barrel are employed underground with "E" rods in 5- and 10-ft. lengths and with cast "E" bits. Underground drilling is done to test the walls and large low-grade areas for ore. The holes generally are less than 150 ft. in depth and are drilled in every direction and at every angle from horizontal. All the cores from the drill holes are put in numbered core boxes that will hold 50 ft. These boxes are stored in the core house, where the core is inspected. The log of each hole is recorded on a log sheet, which also shows the elevation, latitudes, and departures, dates drilled, and general information. Log sheets are kept in a loose-leaf folder in a vault in the main office.

Diamond drilling is the principal method of prospecting and exploration. Results show also that it is the cheapest and most reliable method. During a normal year 10,000 ft. of holes are drilled; the footage usually increases with increase in metal prices. The diamond-drilling cost during the fiscal year 1938-39 for 9,523 ft. was $1.04 a foot. It should be noted that diamond-drill holes are not used to determine the importance of the orebody. Their entire purpose is to direct the development and mining of the ore sections. Ore discovered by underground drilling usually is opened up by a short tunnel or raise from the nearest stope. This is done as quickly and cheaply as possible. The orebodies discovered by underground drilling may prove to be very large or small.

Methods of Sampling and Estimating Tonnage and Value—The only samples from underground that are assayed are diamond-drill cores. If an ore section is at least 3 ft. thick and assays 5 percent or more combined zinc and lead, the orebody in that vicinity usually is minable. It is difficult to estimate tonnages, and several methods that have been applied have failed to give accurate results. The method now used is far from accurate in detail, but it gives a useful overall estimate of tonnage. In a drilled area the greatest dimensions in length and width are multiplied, and the result is the approximate tonnage in that area. For example, an area that is drilled for 1,000 ft. in length and 300 ft. in width will contain 300,000 sq.ft. of surface area. For each square foot of surface area an average of 1 ton of ore is assumed, making an estimated tonnage of 300,000. In areas of 100,000 sq.ft. or less the estimate usually is very inaccurate, but in larger areas the results have approximated actual tonnage surprisingly.

Methods of Development and Mining—The mine is developed by a 1,000-ft., 7x8-ft. tunnel known as the Zero level; by a 300-ft., 5x10-ft., 80 deg. incline known as the Josephine shaft; by a 1,500-ft. 8x8-ft. tunnel known as the 500 level; and by a 320-ft., 5x10-ft. vertical shaft known as the Main shaft.

The collar of the Josephine shaft is at the same elevation as the Zero-level tunnel, and the collar of the Main shaft is at the same elevation as the 500-level tunnel. The Zero tunnel and the Josephine shaft have not been in use for the past five years. At present 80 percent of the ore is hoisted through the Main shaft and 20 percent is hauled through the 500 level. The portal of the 500 tunnel is 400 ft. from the mill, and the Main shaft is 1,200 ft. from the mill. The 500 level was driven to intersect the orebody below the 300 level and was used for a main haulage tunnel for the ore on and above the level. When the extension of the orebody below the 500 level was indicated by surface diamond drilling it was decided to sink a shaft. The shaft was built as near as possible to the center of the orebody, as indicated by the drilling, and at the same time reasonably near the mill. The Main shaft has two levels—the 700 and the 800. The 800 is the main transportation level. It is provided with ore and waste pockets, one on each side of the shaft.

Shaft Sinking—The Josephine shaft was sunk 100 ft. during 1916 and was deepened in 1928 to the 300 level. The Main shaft was sunk during the winter of 1935–36. Rock dimensions of the latter are approximately 12x7 ft. The timber is of fir and larch, 8x8 in., with sets at 5-ft. centers.

Three crews of miners worked in 8-hr. shifts. A crew consisted of two miners, one helper, one top man, and one hoist man. On the day shift a timber man framed and prepared all the timber used. The crews performed whatever operation confronted them, whether drilling, shoveling, or timbering.

A 32-hole V-cut round was used, which broke about 5 ft. in depth. Chicago Pneumatic sinkers were used with 1-in., hollow, hexagonal steel with forged bits. Forty-percent-strength gelatin dynamite was used with electric delay blasting caps, fired from a 220-v. circuit. The broken rock was shoveled into 20-cu.ft. buckets. A temporary headframe was constructed for the sinking operation; during sinking, the main headframe and ore bins were installed.

Drifting—All tunnels in the mine are called drifts. The drifts are 6x7 ft. in section and are untimbered; they are given a 0.5 to 1 percent grade in favor of the loads and are provided with a water ditch on one side. Most drifts are less than 100 ft. long. They are run in ore if possible regardless of the direction they may take. The track is laid in the center of the drift with 40-lb. rails. The main water and air lines are placed on the drift floor and cross under the track wherever necessary.

The following method has now been used on the 800 level for more than 1,000 ft. of drifting:

In the face a crew consisting of one miner and a chuck tender drills a 31-hole, box-type, burn-cut round 9 ft. deep. The round is loaded with 350 1¼x8-in. cartridges of Trojan, 50 percent strength, standard powder. Delay electric blasting caps (No. 0 to No. 9) are fired from a 440-v. line. The round pulls from 8 to 10 ft. of ground, depending on the faults at the bottom of the holes. Standard 3½-in., 7-ft. drill columns are used with 3½-ft. arms. The drills are 3½-in., 180-lb. drifters. Jack bits are used, with ⅛-in. gage change on 1¼-in. hollow round drill steel.

The next shift of two men mucks out the round with a Sullivan overhead air loader. A 2-ton mine car is used and is hand-trammed 800 ft. to the skip pocket. After cleaning the face the crew lays the tracks and pipe.

Drifting costs per foot for 757 ft. were as follows: Miners, $0.97; chuck tenders, $0.72; mucker operator, $0.66; trammers, $0.83; hoisting, $0.30; powder, $0.26; miscellaneous, $0.21; total, $5.95.

Excerpted from information circular No. 8036, U. S. B. M. "Methods and Costs of Mining Zinc-Lead Ores at the Pend Oreille Mines and Metals Co." By Charles A. R. Lambly, March 1941.

Methods and Costs of Mining Zinc-Lead Ores at Pend Oreille Mines & Metals Co., Metaline Falls, Wash.—II

RAISES are run from the drifts to the top of the ore section. These seldom are more than 100 ft. high; they are 10x5 ft. in section and usually run on a 45-deg. slope. Timbering is seldom done until a raise is complete; then most of it is for chute gates only. In new areas raises are divided into a manway and an ore compartment.

Two miners using 120-lb. wet stopers drill a 20-hole, V-cut round, which pulls 6 ft. Jack bits are used on 1-in. hollow, hexagonal drill steel. A round is drilled and blasted each shift. Trojan powder is used in the holes, with a cartridge of gelatin dynamite in each hole to hold in the Trojan powder. The broken rock is loaded by a Sullivan overhead loader or by a scraper until the chute gate has been installed.

Mining Open Stopes With Pillar Support—The limestone formations are favorable to open-stope methods, as the walls and backs stand unsupported without caving. Areas exceeding 100 ft. in all three dimensions have stood opened for ten years without any sloughing from the walls or backs.

After a drift has entered in ore zone a raise is driven on a 45 deg. slope to the hanging wall of the ore. The hanging wall is determined by the metal content; it is generally blank or very low grade. At the top of the raise side-swiping with a 3½-in. drifter is begun, usually all around the raise. The ore is followed wherever it goes. The drifter is mounted on a 7-ft., 3½-in. column with a 40-in. sliding arm and is installed on one side of the raise. This permits mining the ore in any direction. As the face advances the back is scaled up to solid ground. This operation is checked closely to eliminate so far as possible the necessity of returning to the back for scaling at a later date. If the ore goes upward another short raise is started to follow it, or a small shrinkage stope is started with stoper drills. As soon as this overhead ore has been removed the main face is again advanced. After all walls on the level are barren and the back is in good condition a bench is started from the raise. The bench is 5 to 12 ft. high. A 3½-in. drifter mounted on a 90-lb. tripod is used for benching. Two or three rows of holes are drilled along the bench to a depth of 8 to 12 ft. After blasting the ore is scraped into the raise, and a new face is prepared for drilling. The benching operation is continued until the ore stops or the wall of the original stope is reached. If the ore continues beyond the wall a heading is cut under the hanging back into the ore, and a side-swipe face is advanced until the ore stops. The benching, as described, is continued until all the ore has been removed from a stope.

In some areas the ore pitches down into the footwall. Some of these sections are situated so that it is not economical to drive a drift or put in a raise to tap the ore at the bottom. When this condition arises underhand stopes are started; these have been worked very successfully to depths of 40 ft. below track levels. A slushing hoist is set on a timber frame, under which a mine car is placed for loading. A cut is made in the floor in front of the frame, and from there it is carried as a regular benching operation. If water is encountered a Chicago Pneumatic sand pump is used. This type of pump has proved satisfactory for pumping the sand-laden water.

During mine operations the backs are watched very closely. If heavy ground appears a "bar-down man" is called. He is supplied with a 60-ft. ladder, long and short bars, spotlight, and other necessary equipment. Work is never carried on under heavy ground, and a mined-out stope is never left with heavy ground hanging in the back or on the walls.

Blasting — Standard-type, V-cut rounds, side-swipe rounds, and benching rounds are used in stopes. Holes range from 5 to 14 ft., depending on conditions. In wet and well-ventilated stopes standard gelatin dynamite is used, whereas in dry and poorly ventilated stopes Trojan 40 percent strength powder is employed. No. 6 blasting caps and White Bear brand fuses are used in most stoping. Electric delay blasting caps are used in the remaining stopes.

Steel—In all drifting and stoping where 3½-in. drifters are employed, 1¼-in. hollow, round steel is used. In raises and in some stopes where stopers are used, and with the sinkers, 1-in. hollow hexagonal steel is used. Most steel is threaded for detachable bits, which have an advantage over conventional steel, principally because of the difficulty of transporting conventional steel to the stope faces. Costs are approximately the same for the bits, but underground transportation costs are greatly reduced.

Mucking—All broken ore underground is handled by means of Sullivan 15-hp. double-drum, electric hoists and by Sullivan 7½-hp., air and electric, double-drum hoists. The air hoists are used in areas lacking electric power. The hoists are placed wherever desirable in the stope and may be moved many times before the stope is mined out. Under low backs the hoists are held in place by 4- to 6-in. poles placed on the hoist's base and wedged against the roof. In high stopes four holes are drilled in the floor, two on each side of the hoist. Eyebolts are wedged in the holes, and drill steel is passed through the eyebolts over both ends of the hoist.

Power is delivered to the hoists by flexible, rubber-covered cable, hung on the walls wherever possible. In stopes, the cables sometimes are laid over the floor and if close to the face are removed before each blast. A switch box fused for motor protection is placed at each hoist.

Holes are drilled in the face about 10 ft. apart, and eyebolts are wedged in. A ⅝-in. chain long enough to go across a full face is passed from eyebolt to eyebolt. Chain hooks are placed on each end of the chain next to an eyebolt to hold the chain in place. An 8-in., Sullivan, roller-bearing block is hooked to the chain where desired and is held in place by chain hooks; then the pull-back cable or tail rope is put in the block. On the pull-in drum, ½-in., Seal-type, rope-center, wire cable is used, and on the tail rope is ⅝-in. cable of the same type. When a cable breaks the ends are tied together. A cable is used until it is completely worn out. On the average, 1 ft. of cable moves about 10 tons of rock 100 ft., or has a life of 1,000 ton-feet.

Scrapers are of the hoe type.

In most stopes the hoist slushes the ore to a chute. In some stopes timber frames are built over a track, and a hoist is set on the frame. A mine car can be placed under the frame, which has an opening directly over the car. Rails are placed on the incline to protect the timbers from wear. Where the ore is loaded into cars the hoist operator hand-trams the cars to the

main chutes. The distance trammed in stopes is 100 to 500 ft. The slushing operator also sledges boulders in stopes; if there are very large slabs of rock they are drilled and blasted at the end of the shift. Drilling and slushing are not done at the same time in a face or bench, if avoidable, but usally on opposite shifts. Hoists are operated by one man, who cleans out one to three faces each shift, depending on size and location, and averages 50 tons a shift.

Haulage—The main underground transportation system utilizes battery locomotives, scraper hoists, and hand tramming. On the 500 level ore is loaded from chutes into 2-ton side-dump cars. A train of six cars is handled by a Westinghouse 60-cell battery locomotive. The train unloads at the crusher house.

The 20-deg. incline taps the area above the 700 and below the 500 level. All ores mined in this area are slushed or hand-trammed to the incline. A 25-hp. Sullivan hoist is at the bottom of the incline. Cables used are ¾- or ⅝-in. on the pull-in and ½-in. on the pull-back. The scraper is larger than those used in the stopes and moves approximately 1 ton of ore per load. The ore is scraped 300 ft. down the incline and dumped into the main skip pocket. One operator delivers 250 tons per shift to the skip pocket.

On the 700 level hand-tramming is employed with 1-ton end-dump cars. Chutes are installed along the level for 1,000 ft. One to three trammers are employed. They average 75 tons a shift. The ore is dumped into the skip pocket. On the 800 level a Mancha battery locomotive hauls four to six 2-ton side-dump cars to the skip pocket. Chutes are of wood with 36-in. openings. Arc-type steel gates are used. The loader stands at the side of the car to avoid falling rock.

Hoisting—A Sullivan 125-hp. hoist with a 60-in. drum and a ⅞-in. cable is used in the Main shaft. A combination 3-ton self-dumping skip and man cage operates in one compartment. In the other compartment is a counterbalance skip.

The hoist has a lifting capacity of 10,000 lb. at 400 ft. per minute. After the mill tonnage was increased and the Main shaft deepened it was necessary to increase the tonnage hoisted through the shaft. The ore load was increased from 4,500 to 6,500 lb. The weight of the skip plus the ore load then totaled 11,000 lb. The old skip, which weighs 3,000 lb., was installed as a counterbalance, thus reducing the average unbalanced load to 8,000 lb.

Organization—The greatest problem in mining the replacement ores of the Metaline district results from the irregular shape, distribution, and grade of the orebodies. At first the mine heads varied greatly, and much waste rock was broken unnecessarily.

Table I—Summary of Costs

Name: Pend Oreille Mines & Metals Co. Period covered: 1 year.
Tons ore hoisted during period: 227,408.
Mining method: Open-stope-and-pillar.

1. Underground costs per ton ore hoisted:

	Labor	Supervision	Compressed-air drills and steel	Power cost	Explosives	Timber	Other supplies	Total
Development:								
1 In ore	$0.056	$0.010	$0.002	.....	$0.018	.....	$0.057	$0.143
2. In rock								
Mining	0.340	0.035	0.040	$0.010	0.136	$0.008	0.125	0.694
Transportation (underground)	0.081	0.014	.....	0.013	.....	.....	0.027	0.135
General underground expenses	0.020	0.005	.....	0.001	.....	.....	.....	0.026
Surface expense (directly applicable to underground operation)	0.006	0.009	0.002	0.001	.....	.....	0.008	0.026
Totals	$0.503	$0.073	$0.044	$0.025	$0.154	$0.008	$0.217	$1.024

Table II—Summary of Costs in Units of Labor, Power, and Supplies

Name: Pend Oreille Mines & Metals Co. Period covered: 1 month.
Tons ore mined and hoisted: 23,303.
Mining method: Open-stope-and-pillar.

	Mining (Stoping)
A. Labor (man-hours per ton):	
Breaking (drilling and blasting)	$0.37
Timbering and filling	0.02
Mucking	0.18
Hauling and hoisting	0.16
Supervision	0.08
General	0.12
Total labor underground	0.93
Average tons per man per shift	7.95
Labor, percent of total cost	68.6
B. Power and supplies:	
Explosives (pound per ton)	0.86
Total power kw.-hr., kv.-a. per ton	4.72
1. Air compression	....
2. Hoisting	2.48
3. Pumping	0.75
4. Ventilation	0.14
5. Lighting	0.02
6. Scraper hoists	1.33
Supplies and power, percent of total cost	31.4

Table III—Detail of Costs in Units of Labor, Power, and Supplies

Development: Rock is hard; most development openings are untimbered.

	Drifting	Raising
Size of excavation, feet	6 by 7	5 by 10
A. Labor (man-hours per foot):		
Breaking (drilling and blasting)	2.28	(a)3.20
Timbering	....	0.50
Mucking	1.90	0.59
Hauling and hoisting	1.10	2.00
Supervision	0.14	0.20
Total labor	5.42	6.40
Feet per 8-hour man-shift	1.47	1.25
B. Power and supplies (per foot):		
Explosives (pound per foot)	20	17
Timber	....	....
Total power, kv.-a. per foot	400	400
1. Air compression	Water	Water
2. Hoisting	400	400

(a) Most raises not timbered. When raise timbered, special crew.

Moreover, time was wasted in the selection of mining faces each day. This problem was solved by this method:

Two engineers go through the mine each morning at different times inspecting the faces. The mine superintendent also visits the various faces. At noon the engineers and the superintendent discuss the problem presented in each stope. The faces to be mined are marked, either during the morning inspection or after the subsequent discussion. The chief engineer then makes out written orders for each shift boss covering the entire shift. Each miner receives a written order telling him in which stope to work, the location of the round, the number of holes and the type of round to be drilled, the location of the drill to be used, and the name of the man who will assist him. Orders are also given to the mucking-machine operators instructing them in which stopes they are to work, which faces to prepare for mining, and the number of tons of ore to be delivered to the chutes. Trammers, also, receive written orders instructing them from which chutes to take ore and the number of tons to be taken. Orders are prepared for two shifts in advance. About 95 percent of the orders are carried out as first given. The rest are changed to meet unforeseen circumstances. The men go to work as soon as they arrive in their stope, knowing exactly what to do, and no time is lost waiting for orders. The shift boss knows exactly where each man is and the work he is performing.

(To be concluded)

Excerpted from Information Circular No. 8036, U. S. B. M. "Methods and Costs of Mining Zinc-Lead Ores at the Pend Oreille Mines, Merges Co." By Charles A. R. Lambly, March 1941.

Exploration and Sampling of Domestic Deposits of Strategic Minerals By the Mining Division, Bureau of Mines

UNDER the authorizations of Public 117, 76th Congress, 1st Session (June 7, 1939), and appropriations available on Aug. 9, 1939, the Mining Division began exploratory operations Aug. 10, 1939, in seven different areas, on deposits concerning which considerable was already known but which, on account of the subcommercial or marginal grade of the ores, had not been explored extensively by private enterprise. The scope of operations has gradually been expanded since then, and the following report is presented to indicate the amount of work accomplished to May 1, 1941, and to summarize broadly the progress made.

In undertaking the exploratory campaign it was considered unlikely that any large deposits of commercial grade would be discovered, but it was and is vital to ascertain with a considerable degree of accuracy the domestic reserves of strategic minerals, their situation, tonnage, and grade and to devise in advance of crucial need methods for beneficiating and processing those subcommercial and complex ores that could be made to yield important tonnages of strategic metals in an emergency.

In selecting deposits for exploration from many hundreds concerning which some information was or has become available during the course of the investigations, the Mining Division of the Bureau of Mines and the Geological Survey have worked in close collaboration, and in actual prosecution of exploratory operations specialists of the Survey have taken active part as consultants to the Bureau's project engineers.

In selecting deposits for exploration projects, tabular deposits or large mineralized masses are favored because they may be explored systematically and because they afford the best opportunity for blocking out reserves in advance of mining, whereas irregular, small pipes and scattered lenses that must be prospected as mined and mined as prospected are in a class for deferred consideration. The types of deposits upon which work has been concentrated may be divided broadly into three groups, which are not necessarily differentiated geologically. They are:

Partly developed deposits or deposits with some ore in sight or from which ore may have been shipped, further exploration of which promises to reveal additional ore.

Extensive, low-grade deposits, the continuity in depth and average grade of which are unknown because they have not warranted exploration and development by private industry. Although subcommercial under normal conditions, such deposits merit special consideration for strategic purposes as they may be capable of furnishing large amounts of metal should a situation arise in which cost becomes secondary. The fact that they are below commercial grade may be in their favor from the strategic viewpoint, as this will insure their being left in the ground until the ore is required in an acute emergency. Deposits may be subcommercial because of low grade, remoteness and difficulty of access, or metallurgical difficulties in utilization. Metallurgical research is relied upon to develop the most economical methods for treating ores of the last named type.

Deposits relatively small individually but representative of a type occurring in sufficient number within a region or extensive geological formation to contain, in the aggregate, considerable reserves. Exploration of a few type deposits should reveal information as to their habit (and in some instances frequency) of occurrence and order of size that will permit closer appoximation of their aggregate tonnage than has been possible heretofore, and facilitate development if need arises.

One project has been started recently that is in a different category from these types. This project is analogous to wildcatting for oil on a favorable geologic structure. It is believed to be warranted because the formation is the same and the structure and general geological setting are similar to those of a large body of good-grade ore in the same region. In this instance three of six drill holes put down to date have revealed ore. Moreover, as a number of similar structures have been mapped in the region, favorable results would give some promise that these also contain substantial orebodies, whose combined tonnage could be of real strategic importance. Technique developed could be applied later to exploring these structures, the most favorable parts of which are buried.

Expenditures—Under appropriations for "Investigation of Domestic Sources of Mineral Supply," as authorized by the enabling act of June 7, 1939, there has been expended to date and obligated for expenditure to June 30, 1941, $750,000. For "Expediting Production of Equipment and Supplies for National Defense," as provided under Act Public 667, 76th Congress, 3d Session, authorizing the exploration of manganese deposits, the procurement therefrom of bulk samples for ore-dressing tests, the construction and operation of manganese pilot plants, and the procurement of ore for pilot-plant feed, there will have been expended and obligated an additional $370,000 for exploration and procurement of samples and pilot-plant feed.

Summary of operations to May 1, 1941—More than 600 deposits have been examined by the engineers of the Strategic Mineral Examination Section; these have been selected from several thousand brought to the Bureau's attention. The work of this section has involved voluminous correspondence, the careful appraisal of all information received, and the rating of the deposits in order of their indicated promise.

Exploration of deposits in 40 different areas had been undertaken to May 1, 1941, involving extensive surface trenching and sampling, tunneling, shaft sinking or test-pitting, diamond or churn drilling, or combinations of these methods. Of the 40 projects, 2 are on antimony, 5 on chromite, 19 on manganese, 3 on mercury, 2 on nickel, 3 on tin, and 6 on tungsten ores. They cover deposits situated in Arizona, Arkansas, California, Idaho, Montana, Nevada, New Mexico, Oregon, South Dakota, Utah, Virginia, and Washington. In addition, under the special manganese appropriation, "Expediting Production of Equipment and Supplies for National Defense," samples of from 2 to 10 tons each of representative manganese ores have been obtained from 100 separate deposits and shipped to experiment stations of the Bureau of Mines for ore-dressing tests. Twenty-five hundred tons of manganese ore was mined from a large, low-grade manganese deposit at Artillery Peak, Ariz., and shipped for pilot-plant feed to the Boulder City plant now under construction; 3,600 tons have been mined from deposits near Las Vegas, Nev., and placed in stockpile at Boulder City. In addition contracts have been let

for procurement and delivery to the plant of 250 tons from each of three separate deposits in Lander and Pershing Counties, Nev., and negotiations are nearing completion for the mining and shipment of 2,500 tons of ore for pilot-plant feed from the Cuyuna range in Minnesota.

Represenative bulk samples aggregating 50 tons have been procured and shipped from deposits near Chamberlain, S. D., for preliminary testing at the Rolla (Mo.) and Salt Lake City (Utah) Experiment Stations.

On the several exploration projects, there had been completed to May 1, 1941, 72,842 ft. of diamond drilling; 4,700 ft. of churn and rotary-bucket drilling; more than 4,000 ft. of shafts; 2,342 ft. of tunnels, drifts, and crosscuts; 74,528 cu. yds. of surface excavation in trenches and pits; 97,878 ft. of roads and trails necessary for access to the projects; and more than 22,000 samples had been taken, sacked, and shipped to the Bureau's laboratories for analysis.

Appraisal of Results as of May 1, 1941

The following summary sets forth the results of the investigations as they may be appraised at this time. Of 40 projects undertaken, 25 are still active, and therefore final estimates cannot be made. Broadly, however, it may be stated that results have been more encouraging than could reasonably be hoped at the inception of operations. Search for ore is a slow process even when operations are vigorously prosecuted and must be conducted step by step in accordance with knowledge gained by earlier steps. A number of deposits are responding to exploratory work, so that it is believed recommendations may be made soon for their development and for production therefrom to supplement stockpiles of strategic ores.

Antimony

As a result of the two exploratory projects it may be stated that a conservative estimate of the tonnage available in the Yellow Pine district of Idaho is 1,950,000 tons of ore containing 47,505 tons of metallic antimony. Of this amount, 1,750,000 tons, containing 35,000 tons of metallic antimony, is somewhat below commercial grade but could be mined rapidly by open-pit methods in an emergency. One high-grade deposit in this area was discovered as a result of our work, and shipment of high-grade ore containing 44 to 54 per cent metallic antimony was begun late in 1940. A second antimony project in California failed to reveal substantial amounts of ore but indicated the presence of numerous small lenses that in the aggregate might provide several hundred tons of metallic antimony.

Chromite

Five exploratory projects on chromite ores have been undertaken. One of these yielded negative results. One in Oregon indicated an assured tonnage of 150,000 tons of 22 percent Cr_2O_3 grade. A third, in Stillwater County, Mont., resulted in revealing in two separate areas over 1,000,000 tons of minable width averaging 23 percent Cr_2O_3 to a depth of 300 ft. below the surface and 150,000 tons averaging 26½ per cent Cr_2O_3 to a depth of 100 ft. below the surface, respectively. The geology is such that these deposits may be expected to persist to much greater depth than this, and therefore actual reserves are expected to be much greater. Though this ore is somewhat subcommercial in grade, it assures an important reserve for strategic purposes, which has been thought to be important enough by the Office of Production Management and the Reconstruction Finance Corporation that plans are being made to finance immediate development and productive operations. Two other exploration projects on chromite have been started recently, but it is too early to forecast their importance.

Methods and Costs of Mining Zinc-Lead Ores at Pend Oreille Mines & Metals Co., Metaline Falls, Wash.—III

(*Continued from July issue*)

Percentage of Extraction—About 95 percent of the ore is removed during the regular mining operations, and about 3 percent is removed during clean-up operations. The remaining ore is left in the pillars, but it is planned to remove it when the stopes are abandoned.

Dilution of the ore by waste and low-grade ores is approximately 15 percent. This dilution cannot be avoided, as it is due almost entirely to the characteristics of the ore beds, which include sections of waste and low-grade within the commercial ores. These sections are from a few cubic inches up to 100 cu.ft. or more. It is not possible to leave the larger waste sections, as they "tie up" large quantities of ore and leave backs and walls in an unsafe condition. There is no dilution from caving walls or backs in the stopes, raises, or drifts.

Wage, Contract, and Bonus System—A sliding wage scale fluctuates with the selling price of zinc in East St. Louis. The following hourly wages correspond to the mining costs given in this paper: Common labor $0.53; chuck tenders $0.58; operators $0.63; miners $0.68. In all tunnel work a bonus is paid for advances of more than 7 ft. per shift.

Ventilation—The mine is ventilated by natural draft; the air streams reverse from summer to winter on the 500-level tunnel. In summer air enters the Main shaft and the 500 tunnel, rises through the mine, and discharges through the Josephine shaft. In winter air enters through the Main shaft and discharges through the 500 tunnel and the Josephine shaft. During the winter the down draft in the Main shaft causes trouble from the freezing of air and water lines and the forming of ice in the shaft.

Fire Hazards—The surface is heavily timbered, but large areas around the buildings and mine entrances are cleared. Water hydrants and hose houses are situated so that they are readily available for use in case of forest or building fires. All buildings are protected by Grinnell-type sprinkler and fire-alarm systems. In the mine there is little fire hazard.

Safety Methods—A safety engineer is employed, who is in charge of all safety work. A safety committee consisting of one miner, one surface repairman, and a mining engineer, under the direction of the safety engineer, regularly inspects all underground and surface operations. The committee investigates all accidents and makes a full report with recommendations to the management. First-aid and rescue instructions are given to key employees and any others who may desire them. All accidents must be reported at once to the shift boss. Underground and on the surface all State safety rules are observed and improved whenever possible. "Safety First" posters are placed on bulletin boards, and accident prevention is considered in all operations.

Exploration and Sampling of Domestic Deposits of Strategic Minerals By the Mining Division, Bureau of Mines

PART II

(*Continued from August issue*)

Manganese

Estimates made some months ago by the Geological Survey of reserve tonnages of manganese and of manganese metal contained therein, by States, and which broadly indicate the position of the United States with respect to reserves of manganese ore are in the possession of the Bureau of Mines. These estimates necessarily are based to a considerable extent on geological assumptions. The data below are not comparable and relate only to the results of the work of the Bureau of Mines on specific deposits it has examined or explored.

Nevada—As a result of diamond-drilling and surface exploration by the Bureau of Mines during the past six months there have been disclosed in the Las Vegas district 1,280,000 tons of ore containing 312,500 tons of metallic manganese, compared to previous estimates of 600,000 tons of ore containing 180,000 tons of manganese. Diamond drilling is continuing with reasonable expectation of proving still larger reserves.

In Lander and Pershing Counties, Nev., three separate deposits have been explored, in which there may now be estimated 300,000 tons containing 60,000 tons of metallic manganese. Operations are continuing in the expectation of increasing these amounts.

Preliminary work in the Virgin River area in Nevada indicates the probability of at least a million tons containing 100,000 tons of metallic manganese, and in view of the results of this preliminary work it is probable that an exploratory program involving diamond drilling will be justified.

Utah—In the Drum Mountains of Utah diamond-drill exploration has revealed about 200,000 tons of ore containing 30,000 tons of metallic manganese. Operations are continuing in the expectation of increasing these reserves. When operations began it was impossible, from the ore exposures, to estimate any reserves whatsoever. At other properties examined and partly sampled but not explored at depth in Utah there are estimated to be 33,000 tons containing 7,425 tons of metallic manganese.

Arizona—Extensive deposits previously known to exist in Arizona have been explored by diamond drilling, which has served to substantiate earlier estimates of several million tons of material, containing 761,000 tons of metallic manganese, of a grade that may prove commercial and that could be utilized in an emergency. In addition, it is estimated there are 150,000,000 tons containing from 3½ to 5 percent, or 5,000,000 tons, of metallic manganese. This low-grade material is not considered suitable for immediate reserve, however, as it is far below commercial grade, but in dire emergency it could be utilized. In other Arizona districts, on the basis of exploratory work, surface examination, and preliminary sampling, there are estimated to be 121,000 tons of manganiferous material containing more than 10 percent manganese, or approximately 28,000 tons of metallic manganese.

Arkansas—Extensive test-pitting operations in three different areas in the Batesville-Cushion region where work is still in progress have revealed 350,000 tons of wad ore estimated to contain about 80,000 tons of metallic manganese with reasonable expectation that further exploration will reveal about three times that tonnage and possibly more.

California—In California, exploration of one deposit has indicated reserves of 57,500 tons containing 16,000 tons of metallic manganese. Preliminary examinations of 13 other properties indicate a reasonable expectation of 225,000 tons of material containing more than 10 percent manganese, or a total of 63,000 tons of metallic manganese.

New Mexico—In New Mexico, exploratory work on one deposit has indicated reserves of 220,000 tons of ore containing 46,000 tons of metallic manganese.

Washington—Four separate deposits have been explored by diamond drilling or shaft sinking with the result that one small, high-grade deposit was discovered containing an estimated reserve of 11,000 tons of ore containing 5,500 tons of metallic manganese. This is a deposit that is now being developed commercially. The three other properties are estimated to contain 25,000 tons of ore, or 6,250 tons of metallic manganese.

It is essential to bear in mind, when considering the foregoing statistics, that, as was anticipated, almost all of the ore developed is subcommercial in grade and would require concentration or processing of some sort to make it usuable for metallurgical purposes.

Besides those already mentioned, two deposits are being explored in Virginia, but to date there has been no indication of positive reserves. One of these projects was designed to develop a technique for finding buried deposits in geologically favorable structures and has met with some success in that manganese has been found at about 170 ft. below the surface in three of six holes drilled to date.

The manganiferous deposits in the Leadville, Colo., district have been studied, and a summary of this study is contained in Information Circular 7125. As most of the ore in this district is in and around old caved workings and is under water, its recovery would be expensive and would require a large capital investment. In addition to a high mining cost, it would be very expensive to recover the manganese in usable form because of its mineralogical association.

One exploratory project has been started recently in the southwestern section of the Butte, Mont., district, and another in Daggett County, Utah, but work on neither has proceeded far enough to indicate ore reserve possibilities.

Mercury

One surface exploration project involving large-scale trenching and sampling was completed in the Bottle Creek district of Nevada. One small deposit of commercial grade was discovered as a result of this work, and a technique was developed for rapid and economical prospecting of deposits. Exploratory work is proceeding on two other properties, one at Coso Hot Springs, in California, and the other in Pike County, Ark. Neither has proceeded far enough to permit reliable conclusions.

Nickel

One deposit of nickel in Nevada was explored by surface trenching and sampling, tunneling, and diamond drilling with negative results, and one large low-grade deposit is being drilled in Montana. An engineer of the Bureau of Mines is en route to Alaska to investigate the nickel deposits on Yakobi Island.

Tin

Two of the largest known tin-bearing areas in the country were explored, one in South Dakota and the

other in southwestern New Mexico, which confirmed the opinion that no deposits are likely to be found in the United States that can supply any substantial quantity of tin even at prices several times the normal present price.

One deposit near Spokane, Wash., which contains both tin and tungsten, is now being explored by surface trenching. Work has not progressed far enough to permit any conclusions regarding this deposit.

Tungsten

In the Yellow Pine district of central Idaho, in the course of exploring a large low-grade antimony deposit, some high-grade scheelite (calcium tungstate) has been discovered. Seven diamond-drill holes in the deposit at various points over a strike length of 400 ft. have revealed tungsten ore of commercial grade, some of it very high grade. As exploration and development are continuing, it is impossible to appraise fully the significance of this discovery, but it is believed to be important, especially as it can be developed rapidly, being near surface and in an established camp with an experienced operating staff in charge. This deposit is being developed as rapidly as possible by the owners of the property.

One deposit was explored in Pershing County, Nev., which, though revealing some new low-grade ore, failed to indicate the existence of appreciable reserves. One other deposit is now being drilled in the same county that has revealed some new ore, but final conclusions will have to await completion of the exploratory program. One deposit explored in Inyo County, Cali., was revealed to be very shallow and of low grade.

Exploration of a deposit in the vicinity of Shoshone, Nev., has revealed considerable new ore, but as work has not yet been completed, final results are not available. Another deposit is under investigation in Southern Arizona, where some new ore of commercial grade has been revealed.

Excerpted from U. S. Bureau of Mines Report of Investigations 3574. By Mining Division Staff.

Problem of Accident Prevention at Small Metal Mines

DURING THE PAST TEN YEARS the accident-prevention efforts of Federal, State, and insurance agencies, mine associations, individual mine operators, and mine-employee groups have been rewarded by a substantial decrease in the accident rate at metal mines, and certain mining operations have established accident records that compare with the best of those in industries believed to have far fewer inherent hazards than mining. Nevertheless, the accident rate for the mining industry as a whole remains definitely worse than that of most other industries.

Some individual metal-mining companies have reduced accidents so successfully that one or two disabling accidents a year are considered a setback, and it thus becomes evident that the poor showing of the metal-mining industry as a whole is due to the fact that many other metal-mining companies are not enjoying reasonably favorable accident experience. The best accident rates usually are established at large mines because they have the incentives of reduced direct cost (many large mines carry their own insurance), reduced indirect cost, more efficient operation, improved employee and public relations, reduced turnover, and last but by no means least, humanitarianism. With such incentives, accident-prevention efforts are logical, and where the work is done efficiently good results usually follow.

The success realized by a safety program is mainly in direct ratio to the amount of well-thought-out determination to reduce accidents displayed by the management, and the successful accident-prevention program at the large operation usually is based on capable and adequate supervision, employee education and training, and the installation and maintenance of safe practices and equipment. To develop these factors, the accident-prevention program generally embodies some or all of the following:

(1) Training in foremanship. (2) Meetings of employees. (3) Safety committees. (4) Operating codes. (5) Accident investigations. (6) Safety inspections. (7) First-aid training of all employees. (8) Careful attention to engineering design of equipment and installations. (9) Adequate guarding of equipment, walkways, etc. (10) Safety rules and regulations. (11) Enforcement of safety rules and regulations.

Such a program implies the availability of a number of competent men, including supervisors, safety engineers, mining engineers, and skilled workers, who in at least some instances must be paid for time devoted to safety as well as for time spent in their regular duties. It also implies that, although financial returns are to be expected, a considerable sum of money must be expended first to initiate a safety program.

The following information was compiled to compare the accident experience of large-mine operations where accident prevention is being attempted in varying degrees with that of small mines where little or no effort is made to prevent accidents.

Based on underground employment only, Arizona mines were divided tentatively into two groups. Those employing fewer than 50 men underground on all shifts were classed as small mines, and those employing more than 50 men underground on all shifts were classed as large mines. This proved a convenient division, as the former were found to average about 11 underground employees to the mine and the latter about 250. Upon this basis, an average of 64 of the small mines of Arizona employed fewer than 16 percent of the total number employed underground during the period 1934 to 1939, whereas an average of 11 large mines employed more than 84 percent during the same period; yet the small mines suffered 41 fatal accidents, or more than 40 percent of the total fatal mine accidents, and the large mines 60 fatal accidents, or slightly less than 60 percent of the total. In Arizona the accident rate at small mines is far out of proportion to the employment when compared with that at the large mines, and it is probable that the direct cost of accidents per $100 payroll at small mines is about twice as high as that at large mines.

There are, of course, numerous small mines in Arizona that have low accident rates. However, these good records are not always continued over a long period; furthermore, the good records of some small operations are offset by equally bad records of other small operations during the same period. Actually, in a subsequent year the position of some small mines might be reversed as far as the accident rate is concerned. This condition is due to the fact that no safety program, or at least no *sustained* safety program, is in effect at most small mines.

(*To be continued*)

Excerpted from U. S. Bureau of Mines Information Circular No. 7147, by E. A. Anundsen, associate mining engineer, Bureau of Mines Safety Station, Phoenix, Ariz.

The Problem of Accident Prevention at Small Metal Mines—II

► THIS ARTICLE, *the first part of which appeared in the September issue, was prepared to compare the accident experience at large mines where accident prevention is being attempted in varying degree with that of small mines where little or no effort of this sort is made.*

LACK of organized effort to reduce accidents at small mines may be due chiefly to the following: (1) Compensation insurance is paid at a fixed rate, regardless of accident experience; there is no apparent financial advantage in accident prevention. (2) Operating capital is limited, and usually the operator believes that he does not have enough funds to carry on a safety program. (3) The operation of small mines frequently is intermittent, and labor turnover is likely to be high; hence the effect of a safety program may be lost at each shutdown. (4) Small operators cannot afford to employ a man who is informed on safe practices and equipment and capable of carrying out the work ordinarily performed by a safety engineer at large properties. (5) Small mining operations often are conducted by persons unfamiliar with mining, mining problems, or the need for accident prevention; such persons sometimes are forced on the local manager because of relationship to stockholders, but they may be employed through misjudgment by the owner or because they will accept low wages.

The preceding seems to hold little hope for improvement of the accident rate at small mines, yet further consideration of these obstacles indicates that they are not only surmountable but that even a minimum of effort devoted to accident prevention is likely to be rewarded substantially.

Accident-Prevention Dividends

The total cost of an accident includes the costs of compensation, medical services, repairing or replacing equipment or material damaged by the accident, loss in production at the time and place of the accident, lost production until the injured man's replacement reaches top efficiency, and other items.

Unfortunately, at small mines (and also at some large ones) all such costs except compensation and medical service are thrown into the operating cost, so that the operator loses sight of them and fails to realize how much accidents have raised his cost per ton of ore; moreover, in many small mines little or no cost accounting is done. These concealed costs have been estimated to be four or five times the cost of compensation, and unquestionably they aggregate at least two or three times the compensation and medical cost.

At small operations compensation and medical costs usually are disguised in the form of a fixed insurance rate. Many small-mine operators do not know what sums have been paid out to disabled employees and to their doctors, yet the insurance rate depends directly on these payments, and if small mines as a group reduced their accident rate the insurance rates probably would be reduced also.

In some localities dividends are paid by the insurer to the mine operator when the latter complies with certain safety requirements, and in other localities dividends are returned to the operator when compensation claims of the employee and medical claims of the doctor have been sufficiently below the insurance premium paid to the insurer. Thus a small-mine operator has an excellent opportunity to realize financial gain through accident prevention by cutting operating cost, reducing the premium rate of the group, and participating in insurance dividends. (Since this paper was presented information has been received from an insurance agency in Arizona that from April 1, 1939, to March 31, 1940, the percentage of claim loss to premium earned was 105, and during the period April 1, 1940, to Sept. 30, 1940, when a safety director was employed, the figure was reduced to 41.1 percent.)

It is not assumed, however, that this improved accident rate can be attained without effort, and a small-mine operator is likely to be discouraged by visions of paying for a relatively elaborate safety program like that carried on by large mining companies. A safety program designed to fit the small operation must therefore also fit the operator's pocketbook.

Fortunately, the simplicity of the small operation compared with the complexity of the large one usually allows corresponding simplicity in a safety program, although the program's success depends on the same vital factor as that of the large operation — the management's belief (which costs nothing) in safety and its determination to achieve a low accident rate.

The program itself boils down essentially to two constituents—a competent supervisor and safety education of the employees. Supervision refers to that of the person in direct charge of mining, as the mine foreman or shift boss, or both if more than one is employed. Safety education involves, first, instilling into the minds of employees a desire to work safely; second, teaching them how to do it; and third, requiring that it be done and possibly giving them some extra incentive for working safely.

Effects of Poor Supervision

Competent supervision frequently is lacking at small mines; able bosses are not attracted by the low pay and uncertain future at most small mines. On the other hand, employment of incompetent bosses is expensive economy, for not only are accidents likely to increase, but the cost of operation due to poor mining practices and methods also increases. Many current mine operators can point to the harm done to an otherwise promising operation by previous operators who, as a result of haphazard mining, "lost their shirts." Unfortunately, some of the critics are following the same path.

Where mining is concerned it should be obvious that a competent foreman is absolutely essential. The additional money spent to retain the services of such a person is an intelligent investment, as with such a foreman the possibility of reducing the number of accidents is good and the future for such a mine will be brighter. Adequate supervision, although requiring greater initial outlay, is ultimately economical.

Employees may be educated in safety at little expense. They first must be convinced that accidents not only are highly undesirable from their standpoint but also are preventable. A course of training that will vividly show them the consequences of injury already is available in first-aid instruction, and such training has the additional advantage of teaching employees to care for injured fellow workers, thereby enabling them to save lives and alleviate the consequences of injury.

Accident-prevention discussions conducted as part of this training or following it are very effective in reducing accidents, as employees are then in a receptive frame of mind and

readily see why as well as how accidents can be prevented.

First-aid training is available to employees essentially without cost to the mine operator; it is conducted by Federal and State agencies and by private insurance companies. Most of such training now is conducted at large mining operations for several reasons: The operators of large mines request it, excellent cooperation is offered the agency conducting the training, and a large number of employees are easily available for the training. In contrast, the operator of a small mine frequently does not desire the training until he is in desperate straits and often cooperates unsatisfactorily, if at all, so that the agency conducting the training sometimes wastes time and money that could have been spent to better advantage in reaching a larger number of men at large mines. The fact remains that these first-aid training agencies are willing at all times to work with operators of small mines, provided they are willing to do their part and do not expect the cooperating agency to accomplish miracles unassisted.

First-Aid Training for *All* Workers

First-aid training of all employees frequently will produce an immediate and decided reduction in accidents, but this reduction is likely to be temporary unless additional long-range safety activity is carried on. A satisfactory accident rate can be maintained by an active employee safety organization; if the employment is small this organization also may act as a safety committee; and if 20 or more men are employed, a safety committee may be formed to represent the group.

An effective safety committee includes the mine foreman, all other bosses employed on the surface or underground, and employee representatives equal in number to the supervisors. The highest official at the mine (manager or superintendent) acts as committee chairman. If more than one shift is employed, the safety committee may be split into subcommittees on each shift. For example, a mine employs two shifts, and a mine foreman, shift boss, top boss, and mill foreman are employed on the day shift; these men are made permanent members of the safety committee. At a meeting of employees the names of men who will volunteer to serve on the safety committee are submitted to the group, and four employees (three from the mine and one from the surface) are elected to serve. Thus the committee is composed of four employees, four bosses, and the manager or superintendent. A similar committee is formed on the other shift, with the mine foreman and manager or superintendent also included as members of the second committee. Employee members are rotated and draw lots to determine their tenure. If the committee originally has four employee members they may serve one, two, three and four months, respectively, at first, and all subsequent employee members will have a tenure of four months; only one employee member will be replaced each month, and new members will have the guidance of those familiar with committee activities. The committees may meet twice monthly, and all employees can meet once monthly. As committee members are volunteers no expense should be attached to their activities, and the monthly safety meetings of all employees can be made interesting enough to insure attendance of most of them even though the meetings are held after working hours.

Activities of the committee should include some or all of the following: (1) Investigation of accidents—The committee should inspect the scene of each accident; interview witnesses and also the victim (if possible); determine the cause, responsibility, and method of prevention; and prepare and post a report. (2) Safety inspections—Selected members of the committee should make a trip through the mine and surface plant once a month and prepare a report on unsafe conditions and practices observed. (3) Safety rules—Safety rules should be prepared by the committee. These rules should be practical, and provisions should be included for enforcing them. They should be submitted to the employees for suggestions and approval and be put into effect as soon as approved. (4) Safety suggestions—Safety suggestions should be solicited from the men, and the committee should consider and pass on them. The donor should be advised as to the action taken on his suggestion, and if the suggestion is rejected he should be given a reason.

(5) Reports—Accident and inspection reports should be posted and verbal statements on the activities and accomplishments of the committee should be made at the monthly employee safety meetings. At the mine employing only five or ten men, the entire personnel can be included in the safety committee.

Some mining operations have been able to establish reasonably good accident records through good supervision alone, and it must be admitted that good supervision is very important. The best, most consistent, and most easily attained good accident records, however, have been achieved only after the employees have been assigned a definite place in the accident-prevention program. In fact, the employees themselves can be depended on to accomplish a decided reduction in accidents if given a responsible part in the safety program, provided the management will cooperate in their activities. Employee safety organizations in numerous instances have proved reasonable, fair, and helpful, especially when the employees have understood that accident reduction is mutually beneficial.

Although poor methods and unsafe practices remain the greatest cause of accidents at small mines, these are the factors that cost least to correct. However, inadequate, poorly maintained, defective equipment, plus poor maintenance, lack of timber, lack of proper tools, etc., also contribute to the high accident rate at small mines. These factors are important because they may add more to the cost of mine operation than unsafe practices. Limited capital is certainly an obstacle to the possession of adequate equipment, although lack of foresight and unwisely spent capital are among the principal reasons for much of the poor equipment many small mines are forced to use.

Unsafe practices and equipment observed at small mines include:

Shafts and Hoisting Hazards

Unsafe Practices—(1) Hoisting cars on inclines without using drags to prevent cars running back down inclines if cable, cable attachment, coupling, or hoist fails. (2) Rock-spillage accumulations or shaft timber not cleaned down. (3) Men working on inclines below active hoisting operations without protection of bulkheads or other means of arresting a car, or cars, that may break loose. (4) Mining too close to the shaft. (5) Men riding bails of buckets and skips. (6) Skips or cages overcrowded when hoisting and lowering men. (7) Men riding cages, skips, or cars with tools, explosives, and other material. (8) Storage of inflammable material such as gasoline, oils, and other lubricants near the shaft in the hoist house.

Lack of or unsafe equipment—(1) Poor hand brakes on hoists; no emergency brake or device able to hold the load if the brake fails. (2) No safety catches on skips or cages; safety catches not tested at regular intervals. (3) Hoist cables not lubricated or inspected; old, rusty, and possibly defective cables in use. (4) Hoists not equipped with indicators or other safety devices. (5) No gates, railings, or bars at shaft stations. (6) No gates on cages. (7) No derails at top of inclines. (8) Poor shaft or incline signaling systems. (9) Poor power units on hoists, such as salvaged automobile engines or gasoline engines installed so as to allow exhaust gases to enter the hoist room. (10) No fire-fighting equipment on the surface.

(*To be Concluded*)

Excerpted from U. S. Bureau of Mines Information Circular No. 7147, by E. A. Anundsen, Bureau of Mines Safety Station, Phoenix, Ariz.

The Problem of Accident Prevention at Small Metal Mines—III

▶ PARTS I AND II *of this article appeared in the September and October issues, respectively. Accident experience at large mines where prevention work is being done in varying degree is compared with that of small mines where little effort of the sort is made.*

Rock-Fall Hazards

Unsafe Practices—(1) Men working and allowed to work directly under rock known to be loose. (2) Loose rock allowed to remain over manways and in regularly traveled drifts and crosscuts. (3) Loose gouge or vein material in stopes not timbered up or taken down. (4) First consideration to "getting the round in" and not to making the working place safe. (5) "Get the round in or we'll hire someone who can" policy followed by some bosses. (6) Timber not kept up to face in bad ground. (7) Stulls not hitched into walls where possible and stulls poorly blocked. (8) Sets poorly aligned, poorly lagged, poorly blocked, and poorly framed. (9) Inadequate support for filled stopes where collapse of timber would result in filling drifts that are in use. (10) Failure to leave cushion of rock on timber of emptied shrinkage stopes. (11) Inadequate or indifferent supervision. (12) Improper mining method (mining without support ground that should be supported).

Lack of, or Unsafe, Equipment—(1) Miners not provided with good bars and required to pull down loose rock with picks. (2) Insufficient or too small timber furnished. (3) Booms and hangers not furnished for driving drifts in bad ground.

Fall-of-Person Hazard

Unsafe Practices—(1) Throwing steel and other material down manways, damaging same. (2) Failure to report hazardous conditions in manways. (3) Failure to warn miners working above when pulling chutes. (4) Working floors not kept clean. (5) Careless flooring of sets and neglect to nail flooring. (6) Projecting nails and other tripping hazards not removed. (7) Failure to establish firm and safe footing. (8) Safety ropes and safety belts not worn.

Lack of, or Unsafe, Equipment—(1) Continuous ladderways without platforms. (2) Unguarded openings in drifts and stopes. (3) Small manways obstructed by pipe, timbers, etc., making passage difficult. (4) Rungs broken or missing from ladders. (5) Hand rails not provided where desirable. (6) Poor material used for stagings. (7) Insufficient lagging for working floors. (8) Ropes and safety belts not provided where necessary.

Explosives

Unsafe Practices—(1) Opening cases of explosive with metal tools. (2) Using open lights in magazine and in direct proximity to explosive. (3) Transporting capped fuse and explosive together in an open box. (4) Riding on skips or cages with explosive. (5) Slitting primer cartridges. (6) Lacing, bending back, or otherwise causing sharp bends in fuse. (7) Tamping primers and explosive unnecessarily hard. (8) Spitting fuse without trimming for rotation. (9) Cutting off too much fuse when trimming for rotation. (10) Staying too long when spitting. (11) Failure to count shots and report misfires. (12) Returning too soon after blasting. (13) Drilling, mucking, timbering, or doing other work in the vicinity of misfires before disposing of them. (14) Attempting to remove misfired charges by digging them out or pulling them. (15) Caching or hiding excess explosive or explosive recovered from the muck pile. (16) Failure to guard shots.

Lack of, or Unsafe, Equipment—(1) Poorly constructed and poorly maintained magazines located too close to roads, dwellings, or mine buildings. (2) Magazine not provided with safe lighting facilities. (3) Metal tools furnished for opening cases of explosives. (4) Short fuse provided. (5) Improper fuse provided. (6) Poor or not enough sacks or containers furnished for transportation of explosives. (7) Substandard switches and wiring used for electrical blasting. (8) Electrical detonators obtained without short circuited leg wires. (9) Testing equipment for electrical detonators or blasting circuits not provided.

Miscellaneous

Unsafe Practices—(1) Lack of cooperation among employees, particularly between opposite shifts ("Let the other shift do it" policy). (2) Men not trained in first-aid or mine rescue. (3) New employees insufficiently instructed or supervised. (4) Poor housekeeping. (5) Failure to report unsafe equipment, conditions, etc.

Lack of, or Unsafe, Equipment—(1) Miners not provided with hard hats, goggles, or protective shoes or boots. (2) Electrical circuits not properly protected or poorly installed. (3) Ventilating equipment not provided or inadequate. (4) Sprinkling equipment not provided or inadequate. (5) Mine rescue crews or equipment not provided. (6) Mechanical equipment inadequately guarded.

In connection with employees, it is observed further that many of those now working at small mines are totally inexperienced or inclined to overrate their experience; it is an unfortunate fact that frequently too few experienced men are available at a given mine to make possible the "apprenticeship" system formerly used in mines.

The conditions outlined in the foregoing, actually observed in small mines, are indicative of the conditions that establish an accident ratio between large and small mines that definitely is unfavorable to the latter. A study of this list will show nearly all items to be readily correctible, and that to remove most of the factors contributing to the high accident rate at small mines will require little more than a determination to reduce accidents, the cooperation of the men, and a relatively inexpensive program of education. If the unsafe practices alone were eliminated, the small-mine accident rate would greatly decrease, and with improvement in equipment should compare with that of other industries.

Conclusion

Small-mine operators have much to gain by a progressive attitude toward accident prevention, and they can achieve a decided reduction in accidents through the following steps: (1) Adopting the policy that accidents can and will be prevented. (2) Employing a competent mine foreman. (3) Training all employees in first aid. (4) Organizing safety committees and encouraging them to carry on a safety program such as that outlined. (5) Making sure that new employees receive the necessary instruction, training, and supervision to render them safe workmen. (6) Obtaining the assistance of private, State, and Federal agencies engaged in accident-prevention work.

(*Concluded*)

Excerpted from U. S. Bureau of Mines Information Circular 7147.

Softening Water With Non-Metallic Minerals

MANY FACTORS and conditions have a marked influence on the degree of completeness attained in any water-softening process, a few of which are original hardness of water, method of adding softening reagent, temperature, agitation, type of tank, time allowed to reach equilibrium, and influence of impurities. An extended discussion of the importance of each of the enumerated factors is beyond the scope of this paper. The many fine textbooks and the countless journal articles will supply the necessary information for those interested in broadening their knowledge of water purification.

Water-softening methods may be conveniently listed under the following headings:

1. Chemical coagulation.
2. Water softening by chemicals.
3. Base exchangers, inorganic:
   - A. Seminatural zeolites:
     - (1) Treated glauconites (i.e., greensand).
     - (2) Treated bentonitic clays.
   - B. Synthetic zeolites:
     - (3) Precipitated and gel types.
     - (4) Fused types.
   - C. Miscellaneous.
4. Carbonaceous base exchangers (zeolites):
   - (A) Organic resin-base exchangers.
5. Organic anion exchangers.

Chemical Coagulation—Almost all sources of water supply, whether lakes, rivers, ponds, or wells, contain suspended solid matter. The solids may occur as large particles that settle rapidly by gravity or as fine colloidal matter that may remain suspended indefinitely. As the colloidal matter is electrically charged, having absorbed charged ions from the surrounding medium, it is necessary to neutralize the charge on the colloid before coagulation will occur readily. Colloidal matter in water ordinarily is charged negatively; but when the opposite charge is present it is difficult to flocculate the fine particles. In either case it is necessary to entrap the suspended matter in a flaky precipitate by use of a coagulant. In addition to furnishing electrical neutralizing capacity, the coagulant must be capable of forming a highly disperse system of a fine precipitate. As the neutralized particles contact each other they collect into flocs, which entrap the suspended matter and drag it to the bottom. The larger the flocs grow, the better the clarification. Alum is the most widely used coagulant; closely followed by the compounds of iron (ferric chloride, ferric sulphate, and chlorinated copperas), sodium silicate, clays, sodium aluminate, lime, and a host of others less important. There are about 20 such reagents. Which are most suitable for a particular water system may be a matter of opinion and may depend on the nature of the water to be treated. Water used in commercial practice is not a pure substance but a liquid that contains many dissolved impurities, positive- and negative-ion radicals, and buffer salts. It is these impurities, in their varying concentrations, that may influence the behavior of the chemical coagulant so as to make it difficult to attempt to predict how a coagulant will act upon a certain water. The safest course to follow in the selection of a chemical coagulant or combination of coagulants for a water plant is to determine, by experimentation, which one is most satisfactory for the particular water in question. L. C. Billings believes—and he has many supporters—that different waters behave so peculiarly with reference to coagulation that it would be economical to construct and operate a pilot plant before the main plant is designed.

Clays (Kaolin, Fuller's Earth, and Bentonite)—Clays consist of certain hydrous aluminum silicates known as clay minerals, and these include kaolinite ($Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$), dickite ($Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$), nacrite ($Al_2O_3 \cdot 2SiO_2 \cdot 2H_2O$), halloysite ($Al_2O_3 \cdot 2SiO_2 \cdot nH_2O$), allophane ($Al_2O_3 \cdot SiO_2 \cdot nH_2O$), beidellite ($Al_2O_3 \cdot 3SiO_2 \cdot nH_2O$), nontronite ($Fe_2O_3 \cdot 3SiO_2 \cdot nH_2O$), and montmorillonite ($Mg, Ca)O \cdot Al_2O_3 \cdot 5SiO_2 \cdot nH_2O$).

Kaolinite, dickite, and nacrite are known as the kaolin minerals. All have the same chemical composition, their optical properties differ somewhat, and each shows characteristic X-ray diffraction patterns, dehydration curves, and behavior towards dyes. Montmorillonite is a group of clays variable in the composition and properties and the term applies to those clays having natural or latent bleaching characteristics. Fuller's earth and the bentonites belong to this group. Montmorillonite is an alteration or weathering product of volcanic ash and appears to have been formed by the action of water, as it is found in both fresh-water and marine sediments. Beidellite is similar in its properties to montmorillonite, as it also is derived from volcanic ash of high iron content.

Natural clays consist of fine particles in a wide range of sizes. These particles are negatively charged and act as coagulants when added to muddy or turbid waters, provided the water suspensoids are positively charged. The theory of coagulation is discussed further later. The tonnage of clays used in water purification is small, but the demand is increasing yearly. The kaolins and fuller's earth used in water purification are mined principally in the Southeastern section, which includes the area from North Carolina to Florida. However, kaolins are distributed widely and may be found in almost every State in the Union. The best grades of bentonite are found in Wyoming, California, and Mississippi. Clay minerals for water treatment are relatively cheap, as the processing operation is simple. As the clays occur mostly at or near the surface they may be dug with power machinery, dried, broken, or pulverized to size, and bagged for shipment.

Bentonitic clay, a true non-metallic, is a newcomer among coagulants for turbid-water purification. It has already met with marked success and it bids well to establish a permanent place for itself in the field. Besides bentonite, kaolin also has been used by G. D. Norcom, who found the addition of kaolin to low-turbidity waters helpful in reducing their solids content. H. L. Olin obtained excellent results with the swelling type or the sodium-bentonite clay. He found this clay had remarkable colloidal properties, the most conspicuous of which was the large increase of volume due to hydration. It also assumes a highly dispersed state in water, which breaks into a precipitate of soft flakes, and this gelatinous mass is very effective in cleaning up the silt. Bentonite customarily is used in conjunction with alum, the most favorable ratio being 1.4 times as much alum as bentonite. A distinct advantage of this clay is that it is not necessary to control the pH of the solution for good curdling or coagulation. Even a high-acid medium does not adversely affect the type of flocculation. The flocs are more voluminous and settle more rapidly than those produced by alum alone. The bentonites, also, have zeolitic properties, but these are not very important. For high bicarbonate waters, these clays are superior to alum alone because there is not as much conversion of a sulphate into the troublesome sulphate hardness. Saving in alum requirements is one important result.

(*To be continued*)

Excerpted from U. S. Bureau of Mines Report of Investigations 3578, by S. J. Broderick, Southern Experiment Station, Tuscaloosa, Al..

Softening Water With Non-Metallic Minerals—II

PAUL WEIR'S experience with a natural fuller's earth was also favorable. He found fuller's earth improves alum coagulation of waters having a turbidity of 100 p.p.m. or less. A heavy floc is formed, which settles rapidly, entrapping the fine, suspended solids and producing good coagulation. In colored waters the clay forms a higher and more absorbent floc with a pronounced tendency to remove organic color, the amount removed being roughly proportional to the quantity of clay used. He also found a complementary effect of the clay when used in conjunction with activated carbon on raw waters for reducing taste and odor. The combination improves coagulation and reduces tastes and odors to a greater degree than does the use of either substance alone.

C. Alexander has reported some singularly interesting results based on his experience in clarifying a river water for a rayon plant. The requirements of a water for rayon manufacture are very rigid, one essential being that turbidity shall not exceed 0.1 p.p.m. Many combinations of coagulants were tried without apparent success until the mixture of clay and alum was suggested. The flocs produced by this treatment were large, settled rapidly, and reduced the turbidity of the filtered water to 0.05 p.p.m.

J. R. Baylis has investigated a number of clays for their coagulating properties on Lake Michigan water and found the clays helpful in forming a heavy floc, which settled rapidly. His conclusions relative to the use of clays for removal of taste and odor were not favorable, as little, if any, reduction of those disagreeable characteristics of the water was noted.

O. W. Gray used clay as the principal coagulant in a hard water and noticed a substantial reduction in hardness after the treatment. The better coagulation produced by the clay furnished a superior water, both chemically and bacteriologically.

Native alums occur in isolated localities in deposits so small that the mineral has little value. Large quantities of the "filter alum" ($Al_2(SO_4)_3$. 18 H_2O) are made by dissolving bauxite in sulphuric acid and concentrating the solution to crystallize a white solid, having the composition represented by the formula $Al_2(SO_4)_3$. 18 H_2O. Clay also has been used as the mineral to produce alum, but it is as satisfactory as bauxite for this purpose. Alums can be crystallized easily from acid solutions, but the crystallizing conditions must be controlled properly; otherwise alums of variable water content will be obtained. Alums are also obtainable as a byproduct from various chemical processes. For water coagulation the octodecahydrate $Al_2(SO_4)_3$. 18 H_2O has proved to be the most satisfactory compound.

Alum may rightfully be called the universal coagulant, as it is used more widely than any other. The use of alum to purify water may serve to remove color or to coagulate suspended matter, or both, depending on the control of the pH of the water. The first-named function of alum is not well-known, nor is the theory sufficiently advanced to explain color removal. A satisfactory explanation, however, based on experimental evidence, for the dual rôle of alum is given by Theriault and Clark and is aptly summarized by Hatfield. For additional alum investigations one is referred to Norcom, Hoover and Langelier.

Iron Compounds: Ferric Sulphate, Ferric Chloride, Ferrous Sulphate, and Chlorinated Copperas—Iron compounds for coagulation rank second to alum in importance, as they cost less and frequently are quite efficient. They have been used successfully in many water-purification plants and in some instances have given better results than alum. Other instances have been reported in which iron compounds have been used advantageously in conjunction with alum. A few examples of the successful use of iron compounds will be mentioned.

Willcomb has considered four types of iron compounds for coagulation, namely: (1) original lime and iron process, (2) chlorinated copperas, (3) ferric chloride, and (4) ferric sulphate. His conclusions may be summarized as follows: 1. Soft waters (under 50 p.p.m.), moderately colored, react with the four common forms of iron coagulant, provided the raw water contains no interfering organic matter. 2. The lime and iron process of coagulation effectively removes manganese at pH-9.0 to pH 9.3. (A peculiar, unexplained observation made by Willcomb was that the impounded water supplies on the west side of Hudson River, south of Mohawk, were likely to show increasing concentrations of manganese during the first ten years of use.) 3. Coagulation with ferric sulphate adequately removes manganese at pH-9.0 to pH 9.5 and produces an effluent of low color and residual iron content.

Bean, at Providence, R. I., experimented with a wide range of coagulants and finally selected ferric sulphate as the most satisfactory. More recently he has adopted ferric chloride in combination with lime as a better coagulant than ferric sulphate.

Kunz's experience with ferric chloride at the Emporia, Kan., water plant has been most satisfactory, as he reports not only better results but also greater economy as compared to other coagulants.

Marsden C. Smith, at Richmond, Va., has done considerable experimenting over a period of years with various iron compounds as coagulants. He found that by the proper selection and use of coagulants he was able to reduce the quantity used by 67 percent in the period from 1930 to 1935. Alum and chlorinated copperas proved most efficient, and it was interesting to note that alum appeared most economical in warm weather and chlorinated copperas in cold weather.

Ferric-alumina—A unique coagulant is "ferric-alumina," which combines the advantages of alum and ferric chloride into a simple substance. The double compound is made simply by mixing ferric chloride with precipitated hydrated alumina to give a product containing 1 part alumina to 3 parts iron. This ratio may be varied over a wide range to suit any particular condition of water. The ferric chloride in the mixture serves as a protective colloid in peptizing the hydrous alumina. The finely dispersed hydrous-alumina precipitate neutralizes the charges on suspended solids and at the same time agglomerates the small particles. In a short time the colloidal precipitate breaks and large floes of aluminum hydroxide are formed, settling rapidly and dragging down with them occluded fine solid matter. Ferric-alumina has been used rather widely in Europe but has not been so enthusiastically received in this country.

Ferric-alumina was selected as the best coagulant at the East Chicago, Ind., water plant after long experimentation with some of the other coagulants. Alum proved unsatisfactory because the fluctuations in the isoelectric point of the suspended matter

were so erratic as to make difficult proper adjustment of the amount of coagulant from day to day. The addition of sodium aluminate to the alum gave greater efficiency, but the dosages still fluctuated. The alum dosages ranged from 1.5 to 4.5 grains per gallon and sodium aluminate from 0.3 to 1.0 grain per gallon. By the use of ferric-alumina alone, however, good coagulation was produced with only slight variations in dosages—1 to 1.5 grains per gallon.

Sodium Silicate—In 1936 John R. Baylis discovered that when sodium silicate was used in conjunction with other coagulants it gave much better results and may have far-reaching possibilities in the purification and conditioning of water. When sodium silicate constitutes 40 percent of the coagulant the resulting coagulation is rapid and complete if the pH is maintained at 5.5 to 7.5. The discovery of sodium silicate as an aid in coagulation came about when Baylis noted that good coagulation was difficult to obtain in distilled water. Continued persistent effort in which new compounds were tried revealed the necessity of the presence of silicates in solution. Sodium silicate was then tried with marked success. Other silicates may prove just as good, but the cheapness of sodium silicate gives it a decided advantage over other reagents. Further research undoubtedly will uncover many interesting applications of this new agent.

Sodium Aluminate—Sodium aluminate is seldom used as a coagulant except in lime-soda softening. It has a wider market in boiler-water treatment and water softening and will be discussed in more detail.

Sodium Zincate—A. P. Black and coworkers feel that sodium zincate is inferior to sodium aluminate, so it is seldom used as a water coagulant.

Titanium Salts—Recent studies by Upton and Russell indicate that titanium salts may find a place for themselves as coagulants. They comment favorably on the use of these salts and maintain that the floes form more quickly, are bulkier, coagulate over a wider pH range, coagulate better in high concentrations of dissolved salt, and give better flocculation at low temperatures than either alum or ferric sulphate. More experimentation is needed, however, to determine the limitations, if any, of these salts when used on widely variable waters.

Water Softening, Lime-Soda—Lime is obtained by calcining the minerals calcite or dolomite, and the term is commonly used to apply to both quicklime and hydrated lime. Quicklime is composed of calcium oxide (CaO) or is combined with magnesium oxide (MgO) and is capable of slaking with water; whereas, hydrated lime is the dry powder that results from treating quicklime with enough water to form the hydrate $Ca(OH)_2$. Either quicklime or the hydrated lime may be used in water treatment. The specifications of the American Water Works Association stipulate that quicklime should contain at least 85 percent available CaO and hydrated lime 90 percent available $Ca(OH)_2$.

Limestone and dolomite are the materials from which most commercial lime is produced. Minor sources are vein calcite, oyster shells, and unconsolidated deposits of calcareous matter formed from the skeletal remains of marine life. The mining of limestone offers no engineering problem, as most of the deposits have little overburden. At places the overburden may be of such a nature as to contaminate the lime with impurities, in which case the mined limestone has to be specially purified. Purity of limestone is of extreme importance to the lime manufacturer. Silica, alumina, iron, or other impurities can be tolerated only in minor concentrations, inasmuch as these amounts are doubled after burning. The distribution of limestone is wide-spread, nearly every State having some deposits.

Burning Limestone

The process of burning limestone is important, as the degree of burning affects both the chemical and physical properties of the finished product. The quality of the rock also may influence the burning, as hard, dense limestone may calcine more quickly, or at a lower temperature than the more porous material. Furthermore, some limestones have a tendency to crumble during burning and others are easily overburned, with a deleterious effect on the properties of the finished product. To produce a product of uniform quality it is essential to use a uniform raw material and to be careful to burn with controlled time and temperature. In the lime-soda process of water softening the hydroxyl of the soluble lime combines with the magnesium of the hard water to precipitate the magnesium as insoluble magnesium hydroxide.

Soda ash and salt cake are two of the basic products of the chemical industry and are used in large quantities; however, the quantity of soda ash used in water softening is small compared to the total amount consumed by industry. The greater part of the soda ash used in commerce is a byproduct of the chemical industry. Sodium carbonate is found in nature as the mineral thermonatrite ($Na_2CO_3 . H_2O$), as nahcolite ($NaHCO_3$), and in alkali lakes as trona ($Na_2CO_3 . NaHCO_3 . 2H_2O$). The natural deposits of sodium carbonate are confined to the Western States, the Searles Lake deposits of California being the most important and furnishing the greater part of the salt for industry. Since the workable deposits of soda ash are brine solutions, the methods of separation and purification are chemical and not strictly mining, so they will not be discussed any further.

The lime-soda process of softening is more widely used than any other form of softening. A number of industries have been built around this process, supplying equipment, consultation service, and chemicals. The process itself is relatively simple, but in practice, for the most efficient results, both up-to-date equipment and a plant of design suitable for the particular needs of the special water are necessary. Each water is a problem in itself. Expert advice from a competent chemical or sanitary engineer in the selection of equipment will be reflected in a more efficient plant. The methods of softening water by precipitation are many, but the principles on which they operate are the same—the removal of calcium and magnesium by precipitation and filtration. Aluminum, iron, and manganese, if present, are in such small quantities that they will not be considered. Where their removal is important, special methods have been devised, and descriptions of these may be found in the literature. Calcium is precipitated as the carbonate and magnesium as the hydroxide. With few exceptions the reagents used are hydrated lime and soda ash. Caustic soda has been used in place of the soda ash, but its use is limited because of the higher cost. Barium compounds also have been used to some extent to replace soda ash, and barium carbonate is to be preferred for use on waters containing large quantities of sulphate.

The following reactions are characteristic of lime-soda water softening:

Temporary hardness reaction:

$$Ca(HCO_3)_2 + Ca(OH)_2 = 2CaCO_3 + 2H_2O$$

$$Mg(HCO_3)_2 + 2Ca(OH)_2 = Mg(OH)_2 + 2CaCO_3 + 2H_2O$$

Permanent hardness reaction:

$$CaSO_4 (CaCl_2) + Na_2CO_3 = CaCO_3 + Na_2SO_4 (2NaCl)$$

$$MgSO_4 (MgCl_2) + Na_2CO_3 + Ca(OH)_2 = Mg(OH)_2 + CaCO_3 + Na_2SO_4 (2NaCl).$$

A limitation of the lime-soda process is the incompleteness of the foregoing reactions; that is, both $CaCO_3$ and $Mg(OH)_2$ are slightly soluble. The early data on solubility of these compounds in pure water are so divergent that they are of little value in calculating solubility. Recent calculations by Larson indicate the co-solubility of calcium carbonate and magnesium hydroxide to be 20 p.p.m. (all calculated as equivalent $CaCO_3$).

(*To be continued*)

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